HANDBOOK OF MILLING DETAILS

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OF

MILLING DETAILS

COMPILED FROM THE

ENGINEERING AND MINING JOURNAL

BY

THE EDITORIAL STAFF

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PREFACE

This book is a collection of articles that have appeared in the *Engineering and Mining Journal* during the last two or three years under the general head of "Details of Metallurgical Practice," a department of the *Journal* that has been appreciated highly by its readers, many of whom have expressed the wish that a collection in book form be made, which has now been done. The character of this book is in all respects the same as "Handbook of Mining Details," published in 1912, and it is difficult therefore in this preface to add anything to what was said in the preface of its forerunner, wherefore some paragraphs of the latter will simply be repeated.

In the editing of this volume, the work has been chiefly in the selection of the material and its arrangement in chapters. Now and then it has been possible to excise some paragraphs as being unessential and occasionally the phraseology of some articles has been altered a little, the requirements of preparation for the original weekly publication not always having permitted leisurely consideration; but in the main, the articles now presented in this book are as they were given in the pages of the *Engineering and Mining Journal*. However, it has been necessary in a few cases to reduce the size of the engravings.

In making this collection the limitation of space necessitated the rejection of all material that did not pertain to the subjects selected for the chapters of the book, and even so it was necessary to dismiss some of the longer articles pertaining to them, which approached the character of essays rather than being the description and discussion of details. The present compilation covers the publications in the *Engineering and Mining Journal* from November, 1909 to December, 1913, inclusive.

No claim is made that this book is a treatise, exhausting its subject, or any part of it. It is simply a handbook that is a more or less random collection of useful information, being just what passes through the pages of the *Engineering and Mining Journal* in the course of a few years. No special attempt to round out any subject has been made. Yet it will be found that many subjects are well, if not fully, treated.

With regard to the authority of what is to be found in these pages: The matter in the main is merely descriptive of what is done. Nevertheless, there is frequently the injection of opinion and advice. A great technical journal is directed by its editor and is shaped by its editorial staff, but it is essentially the product of its contributors. It is a coöpera-

tive institution and its pages are a symposium of the experiences and views of many professional men. During the three years 1911–13, there were upward of 750 contributors to the *Engineering and Mining Journal*, exclusive of the members of the editorial staff and its regular coadjutors, and its news correspondents. Many of these contributors furnished articles that are now collected in this book. Their articles generally are signed. The unsigned articles are chiefly the work of members of the editorial staff of the *Journal* who have been sent into the field to study metallurgical practice.

The heterogeneous authorship of this book naturally gives rise to some inconsistencies, some differences of opinion and some conflicts in advice. It has seemed to me best to let these stand just as in the original, since they are often merely the reflection of different conditions prevailing in different parts of the country, and if carefully read, absence of unity in this respect will not be misleading.

The title of the book is not perfectly descriptive, inasmuch as besides milling there is something about smelting and refining in certain of its chapters. However, the major part is devoted to milling only and it did not seem good to adopt a long and cumbersome title in order to be absolutely precise.

The selection and arrangement of the material in this book, together with a large part of the revision of it, was the work of John Tyssowski, engineer, formerly a member of the editorial staff of the *Engineering and Mining Journal*, and still a frequent and valued collaborator. Mr. Tyssowski is also the author of many of the articles appearing in these pages.

W. R. INGALLS.

July 1, 1914.

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HANDBOOK OF MILLING DETAILS

I

SAMPLING

The Van Mater Sampler (By John Tyssowski).—The sampling machine shown in the accompanying drawing was designed by J. A. Van Mater, head of the mining department of the New Jersey Zinc Co. The sampler as first installed at the hard-rock mill of the Bertha Mineral Co., at Austinville, Va., was intended for use in a space where none of the standard grades of samplers could operate satisfactorily. In this case only 17 in. of height was available. The sampler worked easily in that space and gave such satisfactory results that several have since been installed by the company.

The idea is extremely simple. A second conveyor is arranged to run below and extend beyond the end of a belt conveyor so as to cut the stream of falling ore from the upper belt. On the original apparatus the sampling conveyor was constructed of two parallel No. 25-link 1-in. pitch belts carrying a single bucket. On later samplers No. 45-link belt with about $1\frac{5}{8}$-in. pitch was substituted. The sprockets in each case were 12 in. pitch diameter. The buckets or sampling troughs, of which any desired number may be used, must be as wide as the stream of ore delivered from the belt, and also must be deep enough to contain all the ore intercepted, as will be explained. The details of the sampling troughs are shown in Fig. 1 in plan and section at *C*. The length and number of the buckets are determined by the size of sample desired and by the number of cuts to be made per minute. The bucket is fixed at either side to the chains of the sampling conveyor, and is provided with trolley wheels which run upon $2 \times 2 \times \frac{1}{4}$-in. angle irons used as tracks or guides. These tracks serve to take the impact when the bucket fills. As shown, a wooden bearing block is provided on the bottom of the bucket; this is pointed so as to offer a minimum resistance to the falling stream of ore through which the bucket must pass on its back journey, and protects the bucket from wear.

As stated, the length and depth of the sample bucket, as well as the number used, depends upon the size of sample desired. Contrary to what might be expected, the percentage cut by the sampler is absolutely independent of the relative speed of the conveyor belt and sampler, or

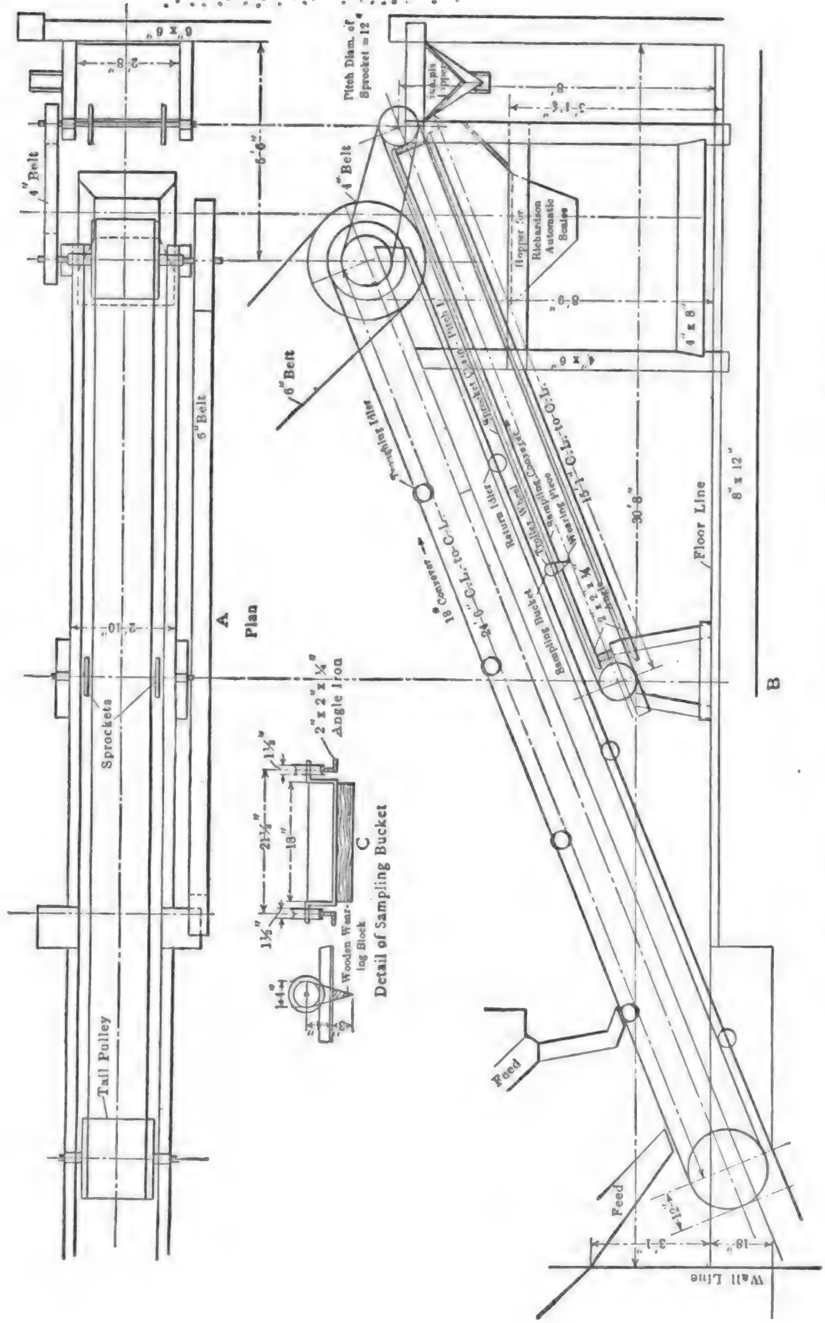


FIG. 1.—CONSTRUCTIONAL DETAILS AND METHOD OF SETTING UP VAN MATER SAMPLER.

the number of cuts made by the sampler. The only effect of making a greater number of cuts is to give a better average sample, within certain practical limitations. To understand this, assume the sampler a solid belt. Then irrespective of the speed of either conveyor, the entire discharge from the upper belt will be caught upon the lower one, and delivered as a sample. If, however, in place of a solid belt for the sampling conveyor, one is substituted in which half of the total length of the sampler is made up of buckets and half left open, then it is evident that one-half of the discharge from the upper belt will be cut as a sample. Similarly, if any desired proportion of the total length of the sampling conveyor is made up of a bucket, or any number of buckets, that proportion of the total stream delivered by the upper belt will be delivered as a sample. Hence the total length of the bucket or buckets with respect to the length of the sampling conveyor alone determines the proportion of the discharge from the upper belt taken as a sample. The respective speeds of the two belts, the number of cuts made by the sampler, or the number of buckets into which that portion of the sampling conveyor which is to catch the stream from the upper belt is divided, only affect the quality of the sample. The object of making several cuts a minute is to get a more average sample, and this can be accomplished by using several buckets or by running the belt at a greater speed. The buckets, of course, must be the full width of the stream delivered from the upper belt and also must be large enough so that all of the ore, falling into them, will be retained.

In the design shown, the sampling conveyor is about 33 ft. long, and the bucket 4 in. long, a proportion of 100:1; therefore, a 1% sample is delivered. As installed, the sampler is run so as to make six cuts per min., this being regarded as giving a fair average sample. Running faster, it was found that too great an amount of the ore was spilled.

(By Franz Cazin).—I have used the sampler above described in a number of mills for obtaining a sample of a passing stream of ore, and have found that on account of its simplicity and the reliability of the device for obtaining a sufficiently correct sample for mill purposes, together with the little head-room required for its installation, the sampler has given entire satisfaction.

The objections which may be raised against the arrangement as shown in Fig. 1, viz., that the bucket may not empty itself completely into the sample hopper and that some particles of the sample may drop into the reject, may be overcome by making the sample hopper sufficiently long and by leading the bucket on its return under the reject hopper (or automatic scale), instead of over it, and over a floor on which the drippings would fall and from which they can be brushed up and added to the sample at the end of each shift.

I used this sampler in 1901 in the Santa Cruz concentrating mill, Santa Barbara, Chihuahua, Mex.; in 1902, in the Penobscot cyanide mill near Deadwood, S. D.; in 1906, in the copper-concentrating mill of the Penn-Wyoming Copper Co. at Encampment, Wyo., and in other instances. So I do not think that this device can be patented now.

I may call attention to a still simpler form of this sampler which I used in the concentrating mill of the Pitt Ores Co. near Breckenridge, Colo. In this sampler I replaced the two chains by a belt to which the bucket is fastened. This sampler can be easily made at any mill.

A Battery-feed Sampler (By J. H. Oates).—The accompanying sketch, Fig. 2, illustrates the hopper portion of a battery feed sampler, which is used successfully at Guanajuato, Mex. In operation, this box is attached to an arm of 1 × 3-in. iron, as shown. This iron, of any required length, is connected and braced to a vertical shaft mounted on a step bearing, which is revolved by any suitable mechanism. In one mill the

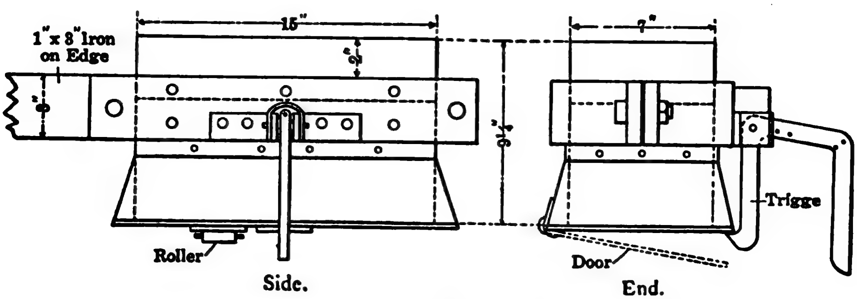


FIG. 2.—BATTERY-FEED SAMPLER.

device is used on a radius of about 6 ft. and is run at about 2 r.p.m. under which circumstances about $\frac{1}{65}$ is taken out as a sample.

The device consists essentially of a box with a hinged bottom, held in place by a trigger. As the box swings under the stream of ore being fed to the mill, the stream is caught in the box. Then, as the box travels over the desired receptacle for the sample, a suitable tripper releases the trigger and causes the weight of the sample to open the door, thereby dumping the sample into the receptacle. The apparatus can be cheaply made in any mine machine shop and requires almost no attention, except to change the upper half of the box as it wears down. It can be used where there is but little head room.

The box proper is made of $\frac{3}{16}$-in. sheet iron, and surrounding it at the bottom is an outer box with flaring sides, as shown, the object of which is to keep the fines from leaking out around the contact of the door with the bottom of the sides of the box. This outer box is made of $\frac{1}{8}$-in. sheet iron. The trigger is of $\frac{1}{2}$ × 1-in. self-hardening steel. A small

roller is put on the bottom of the box. After the sample has been discharged, this roller engages a guide rail, which lifts the bottom again to place, when the trigger automatically closes upon it, holding it tightly in position. It is well to have the trigger catch slightly hollowed so as to engage on a small piece of half-round steel on the bottom. This is necessary, as otherwise the ore on striking the bottom will jar the catch loose and spill the sample.

Sheridan Oscillating Ore Sampler.—Leslie M. Sheridan, of Anaconda, describes a new oscillating ore sampler in U. S. patent No. 1,031,385. The main features are the lightness of the moving part, which is exposed to wear, and the ease of replacing it in event of injury. Referring to Fig. 3, the device consists essentially of a cutter *A*, oscillating in a sampling hopper, which is divided into two longitudinal sections, each of which has a separate delivery spout. When the cutter *A* is at either end of its oscillation, the ore flows through spout *B* down into section *C*

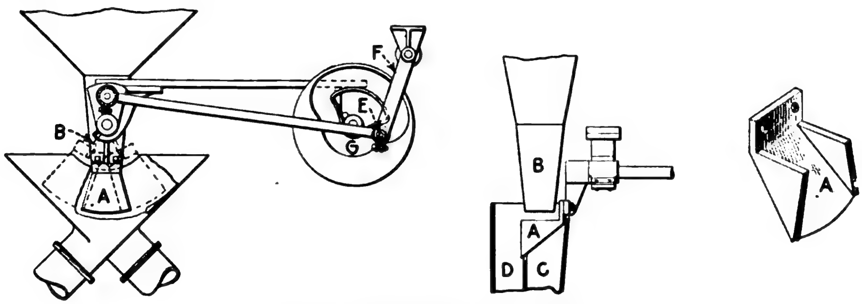


FIG. 3.—SHERIDAN SAMPLER.

of the sample hopper, and on to the ore bins. When the cutter *A* swings under spout *B* the ore is deflected to the sampling apparatus proper. The patent also covers a peculiar cam motion, which regulates the relative periods during which the cutter is under the spout and when it is not. This motion is the resultant of the motion given the rocker arm *F*, because of the pin *E* having to follow the cam groove *G* as the wheel rotates.

Automatic Sampling at Tigre Mill.—The distributing and sampling device, used at the Tigre mill, El Tigre, Sonora, Mex., is described by D. L. H. Forbes (*Bull.* A. I. M. E., August, 1912). It is used to distribute the ore between its two mills and at the same time to take a 5% sample. The device, shown in Fig. 4, consists of a cast-iron cone with a single discharge spout that revolves about a vertical axis over a stationary cone with three compartments. Between the two large compartments is an adjustable partition which may be set to various positions corresponding to percentages of the total ore that it is desired to give to the No. 1 mill. From the compartment corresponding to No. 2 mill, a discharge spout

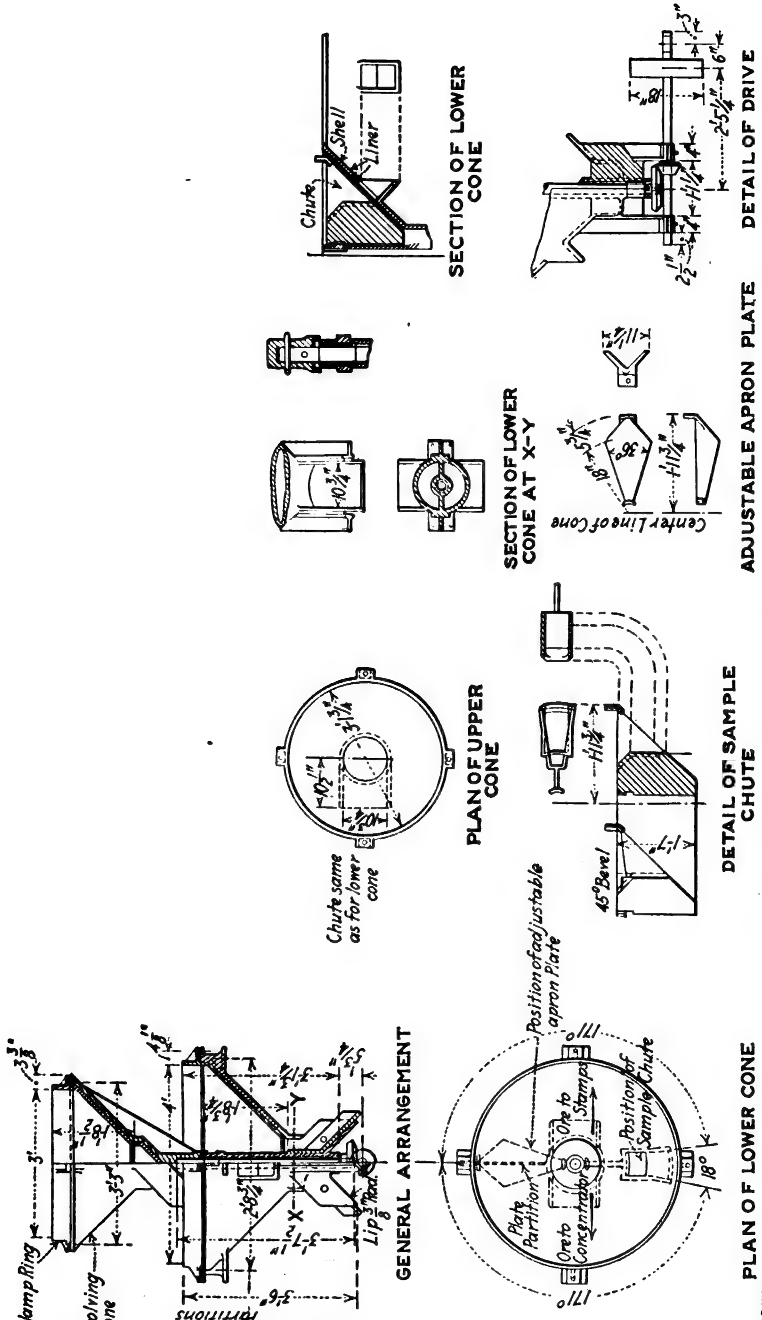


FIG. 4.—AUTOMATIC ORE SAMPLING AT TIGRE MILL, EL TIGRE, SONORA, MEX.

delivers the material to a 12-in. conveying belt, which carries it to the stamp battery ore bin. From the sample compartment the portion of ore cut for sampling is dropped into a 7 × 10-in. Blake crusher, which reduces the size of the largest pieces to 0.75-in. ring. A Snyder sampler placed below the crusher takes out 10% of the original sample, and the large pieces in this sample are crushed in a 2 × 6-in. Sturtevant roll-jaw crusher, after which the amount of the sample is reduced again by sets of riffle boxes until about 40 lb. are obtained for sending to the assay office. The rejected portions of ore at each step in the sampling drop into the broken-ore storage bin of No. 1 concentrator.

An Auto-hydraulic Sampling Device (By D. A. McMillen).—The sketches, Figs. 5 and 6, show in plan and elevation an easily made and

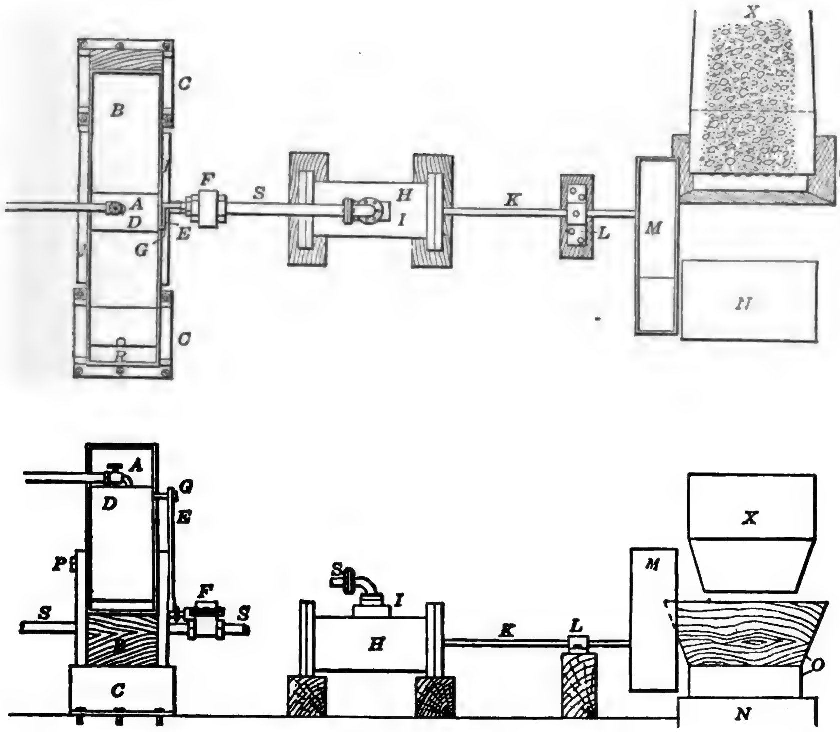


FIG. 5.—TOP AND SIDE VIEWS OF AUTOMATIC HYDRAULIC SAMPLING DEVICE.

accurate sampler, that is used several places in the Globe district and that any mechanic can construct at the mill. The essential parts are a tipping box *B*, a small steam or air cylinder *H* and a sampling arm *M*. The tipping box is mounted on a stand *J* so that it may easily turn about the supporting bolt *P*. The box is restrained in its motion from side to

side by the small stands *C*. In the middle of the box is built a partition *D* extending some distance above and dividing the box into two separate units. A stream of water is directed into the box at *A* so that it will fall into one compartment at a time. When sufficient water has run into one compartment of the box it becomes overbalanced and tips over, exposing the other compartment to the stream of water. The water in the first compartment then runs out of a hole *R* in the bottom of the tipping box. A backward and forward motion of the tipping box, which can be regulated and timed to almost any limits by increasing or decreasing the stream *A*, is thus established. Bolted to the upper part of the divider *D*, by bolt *G*, is an arm *E* extending down to the throttle of a quick-acting valve *F*. This valve controls a steam or air line *S* which is connected with the cylinder *H* through the valve *I*. When the box *B* has enough water in one compartment to cause tipping, the rod *E*

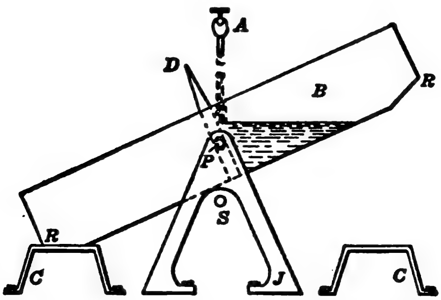


FIG. 6.—TIPPER ON HYDRAULIC SAMPLER.

pulls over the lever of the valve *F*, admits steam into the valve *I* and this throws forward or back the piston in *H*. The piston rod *K* is connected with a sampling arm *M* which is pushed across in front of and below the end of the belt conveyor. A portion of the load of the belt falls into the arm *M* as it passes and is diverted into the sample box *N* instead of falling directly into the bin opening *O*. A bearing for the piston rod is indicated at *L*. The arm *M* makes only one trip across the belt for each tip of the box and thus rests on alternate sides of the belt conveyor as alternate sides of the box are filling. The top of the dividing partition *D* may be weighted to any extent to allow of a higher or lower water level in the tipping-box compartments.

Sampling Roaster Feed.—At the Kalgurli gold mine, Western Australia, the sulpho-telluride ores are fed to the roaster by a short screw conveyor. Through a half-inch hole in the bottom of the casing surrounding the screw a stream of ore is delivered on the point of a cone, placed close up to the casing. A chute attached to the bottom of the cone diverts a portion for assay, the remainder being collected and re-

turned to the fine-ore bin. F. G. Brinsden says that various methods of sampling have been tried, and comparisons of gold contents and grading analyses show that this is a reliable method of arriving at the head value

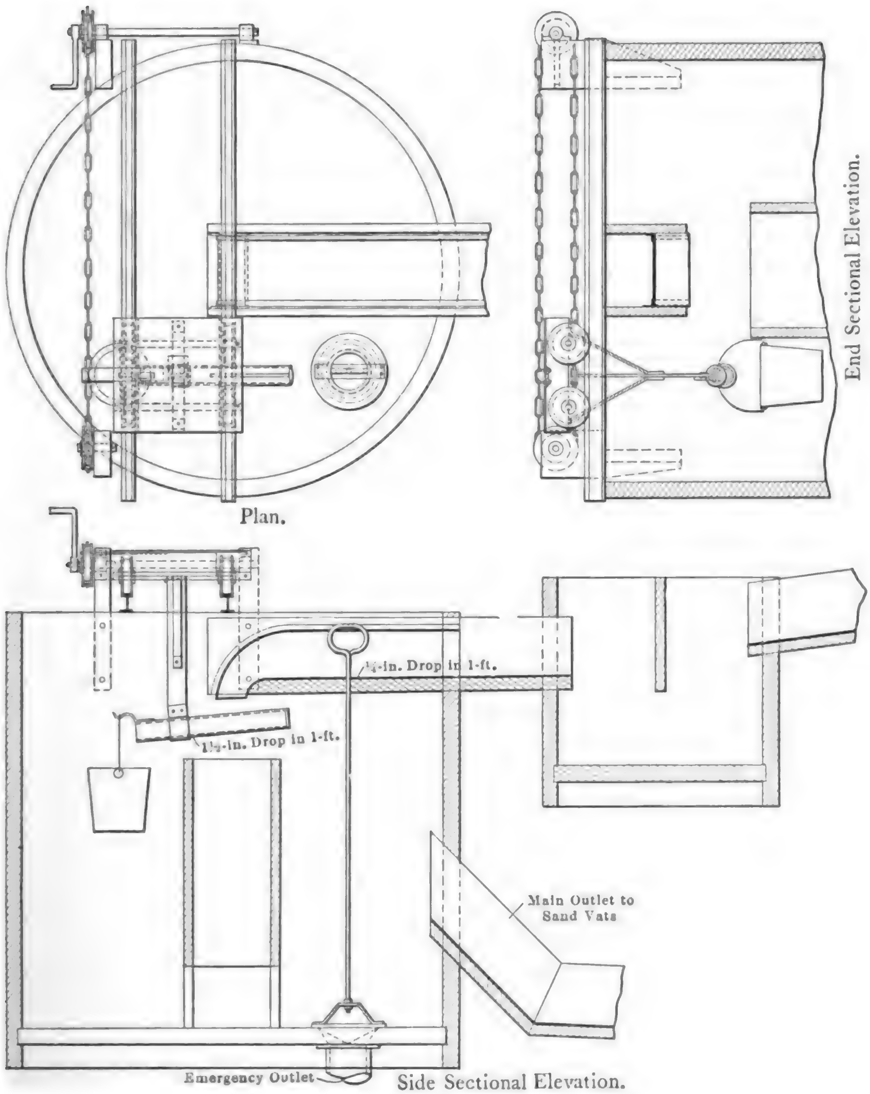


FIG. 7.—HOMESTAKE SAND-SAMPLING DEVICE.

of the ore. There is rarely any exceptional discrepancy between the actual and theoretical gold returns.

Sampler for Cyanide Plants.—The sampler shown in Fig. 7 was de-

signed by W. H. Todd, and has been in use for a year at the Homestake sand cyanide plants in the Black Hills. The features of the design to which particular attention was given are: A small fore-sump, provided to receive pulp from several converging streams, and to cushion the velocity of these streams, thus insuring a smooth and steady discharge of pulp to the sampling sump; an iron deflecting plate at the discharge end of the short launder, connecting the fore-sump and the sampling sump, to direct the stream sharply downward as it delivers the pulp upon the slotted sampling pipe, preventing splash and possible contamination of the sample, for which further protection is provided by a short open conduit, placed vertically and immediately below the sampling blade. The discharged pulp is in this manner conveyed quietly to the bottom of the sump, whence it passes to the leaching vats. The slotted sampling pipe is brought to a sharp cutting edge on its upper face, and is sufficiently inclined to insure clearance of all particles of the sample. It is supported from a carriage resting on overhead rails and is actuated by a chain driven

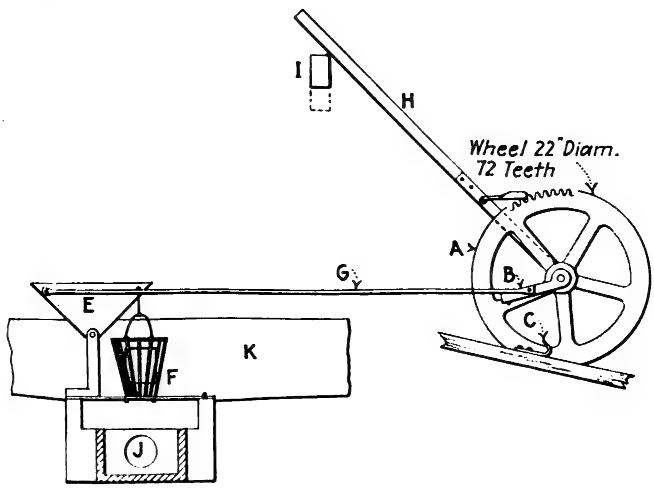


FIG. 8.—AUTOMATIC SAND SAMPLER.

by a winch, which in the sampler illustrated is actuated by hand, but which might as readily be attached to any convenient source of power. By this arrangement the sampler is caused to move across the stream at a uniform speed, and with the slot always at right angles to the falling stream.

Automatic Sand Sampler (By L. D. Davenport).—The automatic sampler described below is used in front of the duplex Dorr classifier in the Ernestine mill, Mogollon, N. M. The power which moves the sampler is derived from one of the classifier rods *I*, shown in section in Fig. 8. This rod moves up and down about 4 in. with each stroke of

the classifier. The top of the rod, protected with a piece of round iron, moves the lever *H*, the lower end of which is supported by the shaft carrying the gear *A*. There are two dogs fastened to the lever *H*, one being about ½ in. longer than the other so that with each movement of the lever, the gear is turned half the width of a tooth. In front of the gear *A*, and turning on the same shaft, is a piece of 1½-in. square iron 9 in. long, marked *B* in the sketch. About 4 in. from the end of *B* is a stud which carries the connecting-rod *G* which in turn is connected to the triangle *E*. This triangle is supported at its lower corner and

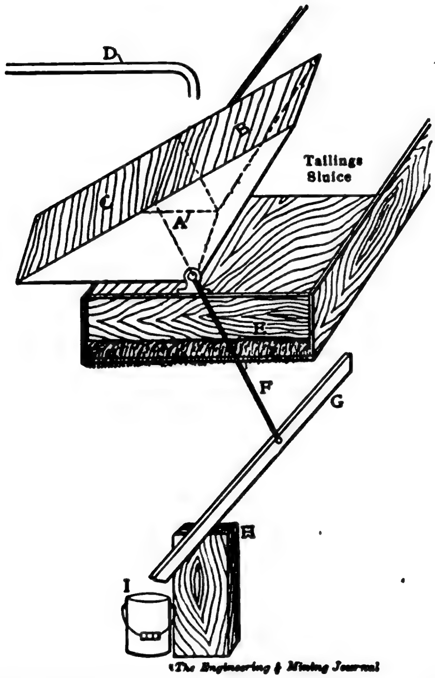


FIG. 9.—TAILINGS SAMPLER AT BUNKER HILL MILL.

the other corner is connected with an iron link to the sampler *F*. This sampler consists of eight half-round troughs 8 in. long by ½ in. deep, fastened together at their ends and hinged at the lower end on a ½ in. iron rod, as shown. The square outlined in the sketch behind the sampler is an opening in the launder *K* through which the sand product from the classifier flows into the launder *J* and thence to the sand vats. The stud *L* on one of the spokes of the wheel raises *B* until the latter stands perpendicular, then at the next movement of the wheel, *B* swings of its own weight through a half-circle, and is caught by the spring *C*, which prevents it from oscillating. When *B* swings through the half-circle just described, the rod *G* transmits the

motion to the sampler F through the triangle E and causes F to dip up a small quantity of sand. The sample thus taken is thrown into a small pan placed directly in front of F. This device takes a sample every 10 min., the total sample for one shift amounting to about 5 lb. It was designed and constructed by Fritz Aude, mill superintendent for the Ernestine Mining Company.

A Simple Automatic Sampler (By Algernon Del Mar).—The automatic tailings sampler depicted in Fig. 9 is simple and easily made. The triangular prism A is of wood or tin, and divided into the compartments B and C. It is pivoted above the tailings discharge E. The rod F is rigidly attached to A so it will move the slotted tin conveyor G back and forth across the stream of tailings. G is pivoted on the block H and discharges into the bucket I. The operation of the machine is as follows: The jet of water which regulates the frequency of the samplings comes through the pipe D, and when one of the compartments C or B has filled sufficiently, it overbalances and swings the slot G across the whole width of the sluice, the sample running through to the bucket. It is now in position to be moved again in the opposite direction when the opposite compartment has filled enough to overbalance. By regulating the flow of water at D the time interval between samples may be regulated.

Improved Teeter-box Sampler.—In Fig. 10 are shown the details of construction of the teeter-box samplers that are to be used to sample tailings at the mill of the St. Louis Smelting & Refining Co., in the southeastern Missouri lead district. This apparatus was designed by G. A. Overstrom. This teeter sampler is conspicuous in the refinements that have been added so as to insure the obtaining of a perfect sample. An important detail of this design is the use of retarding plungers and cylinders which so regulate the tilting of the teeter-box as to insure the sampling spout traveling evenly through the whole section of the stream of tailings.

In the tailings feed box A, the baffles B serve to break up the surface currents, while the feed is drawn off through a slot in the front of the feed box so as to insure an even distribution of the overflowing tailings. To the front of the discharge spout is fastened an angle iron that extends up high enough to dam back a layer of tailings on the spout and cause the tailings to discharge with a slight upward swirl at the bottom so as to mix the heavier particles with the feed. The depth of the issuing stream can be regulated by the angle iron.

This sampling spout D is made 1 in. wide so as to insure that it will take a reliable sample. The sides of the spout extend above the top of the pulp stream so that there can be no overflow and loss of some of the slime. This is an important point that is not always considered in

building teeter-box samplers. Below the sample spout, and on the near side of the catch box is nailed a piece of sheet iron *E* arranged to prevent any splashing of the tailings. The pulp from the catch box is drawn off into the tailings launder through a slot *F* in the side of the catch box.

The sampling spout is carried on the end of a long iron arm extending down from the teeter box *H*. This is supported from two timbers by the

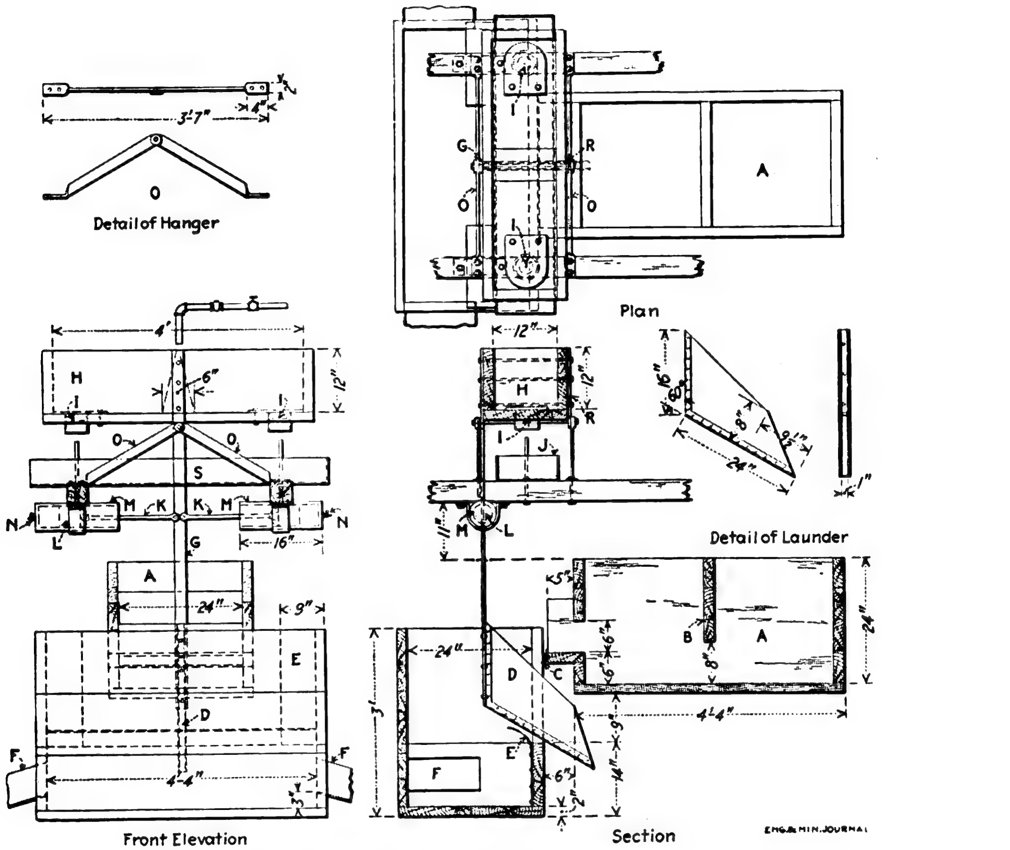


FIG. 10.—IMPROVED TEETER-BOX SAMPLER.

iron rests *O* carrying the bolt *P* which goes through both the hole in the spout arm *G* and that in the strap *R*. The strap is bolted to the teeter box by three bolts passing through the block that serves as the middle partition in the teeter box. In each compartment of the teeter box is placed a valve *I*. This, as the box falls, drops on the pin *J*. This pin then lifts the valve and lets the water run out of the box. The water discharges into the launder *S*. Linked to the spout arm are the two plunger rods *K* that connect with plungers *L*, sliding in the cylinders *M*

which are strapped to the underside of the cross beams that carry the supports *O*. On the ends of these cylinders are the pet cocks *N*. It is by the amount that these pet cocks are opened that the speed with which the spout cuts through the stream of tailings is regulated. The frequency with which the sample is taken depends upon the stream of water that is fed to the teeter box.

Flood Automatic Sampler.—The Flood automatic sampler is being much used in Colorado mills and, as the device is simple and easily placed and cared for, its use seems to be on the increase. It consists of a sheet-metal spout by means of which the sample is cut from the stream of pulp or ore and directed into a proper receptacle. This spout is sustained and moved by an arm which is actuated by a solenoid. An eight-day clock controls the movement of the sampler by closing a knife switch

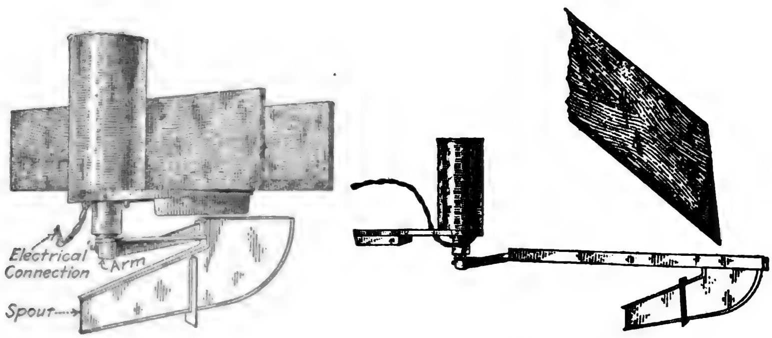


FIG. 11.—FLOOD AUTOMATIC SAMPLER.

at desired intervals, the switch remaining closed for from 1 to 5 sec. as required. When the switch is closed the current reaching the solenoid moves the arm and sample spout into position under the stream to be sampled and keeps it there for the required time. When the switch is opened the spout returns immediately to its normal position.

The cut showing the device, Fig. 11, is self-explanatory. Referring to the left-hand illustration, the hole in the bottom of the launder is encircled by the casting which is part of the sampler, the whole being self-contained and needing no adjustment. The stream passes through this hole at all times and is sampled at the desired intervals as explained. One clock will control any number of machines and can be installed in the office or any other place where it can be under guard. The machine can be made in the long-arm style, and is appropriate for sampling dry-crushed ore or pulp.

A defect in the machine, which may, however, be easily remedied, is that the receiving part of the spout is generally made too narrow. When

the spout is in the stream of material it can receive only a portion of the fall, whereas to be most satisfactory the whole stream should be taken for the given space of time.

Due to the fact that the machine is governed by clockwork and electric current, assuring that a sample be taken at stated intervals, and that the time during which the sample is taken is always the same, a correct sample should result. It may be so accurate that the total tonnage of ore or pulp passing per day could, with suitable checking, be calculated from the weight of the sample, the intervals and time of taking sample being known. The machine is patented and is made by Hendrie & Bolthoff Mfg. & Supply Co., of Denver.

The Borcherdt Automatic Sampler for Sands and Slimes (By W. O. Borcherdt).—From time to time there have appeared in the (*Engineering and Mining Journal* descriptions of various forms of mechanical samplers, especially intended for the sampling of sands and slimes, such as the feeds and tailings of sand jigs, tables and vanners. Of those that have come to my notice usually some defect, such as too great complexity or inability to take an accurate sample, has made them impracticable for ordinary mill use, where simplicity, combined with effectiveness, is essential. All these devices have one point in common, their reliance upon outside power to cause them to function, whether it be applied from the mill shafting or through the medium of a water supply and the familiar tipping trough. As it is frequently desired to use an automatic sampler at a point in the mill, or its environs, where power is not readily available, a sampler should prove useful, which, besides combining the elements of simplicity, accuracy and sturdiness, is also independent of any power other than that derived from the pulp flowing through it. The machine shown in detail in the accompanying illustrations meets the above-mentioned conditions, and during a year's operation has required little attention and no cost for repairs or renewals.

Referring to Fig. 12, showing the general arrangement, it will be seen that the single moving part of the sampler consists of a wheel composed of a sheet-steel rim and a wooden center, or hub, joined to the rim by a number of equally spaced sheet-steel vanes, or blades, extending from side to side in the plane of the axis. In the sampler illustrated, No. 18 U. S. Standard sheet steel is used for the vanes which are radial and are secured in saw cuts in the hub merely by friction, while their outer ends are locked together and to the rim by means of wooden lock strips, with sides cut to the radial bevel, and driven in simultaneously so as to preserve correct spacing and the circular contour of the rim. At *B* in Fig. 13, is shown the butt joint of the rim. Strap rivets are countersunk on both sides. This joint should counterbalance the sample bucket.

The number of open cells thus produced is suited to the size of the

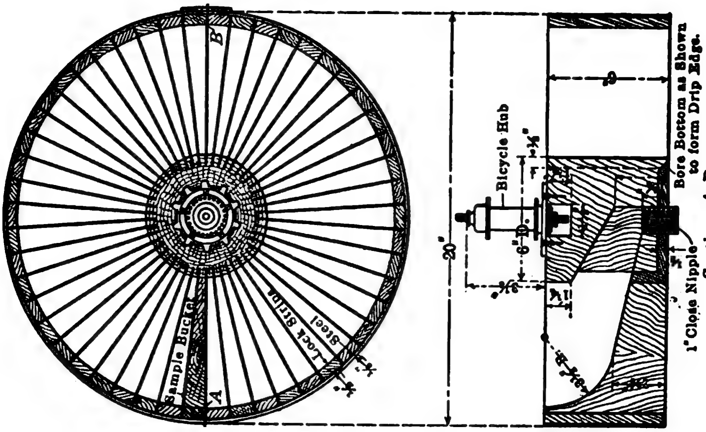


FIG. 13.—DETAILS OF SAMPLER WHEEL.

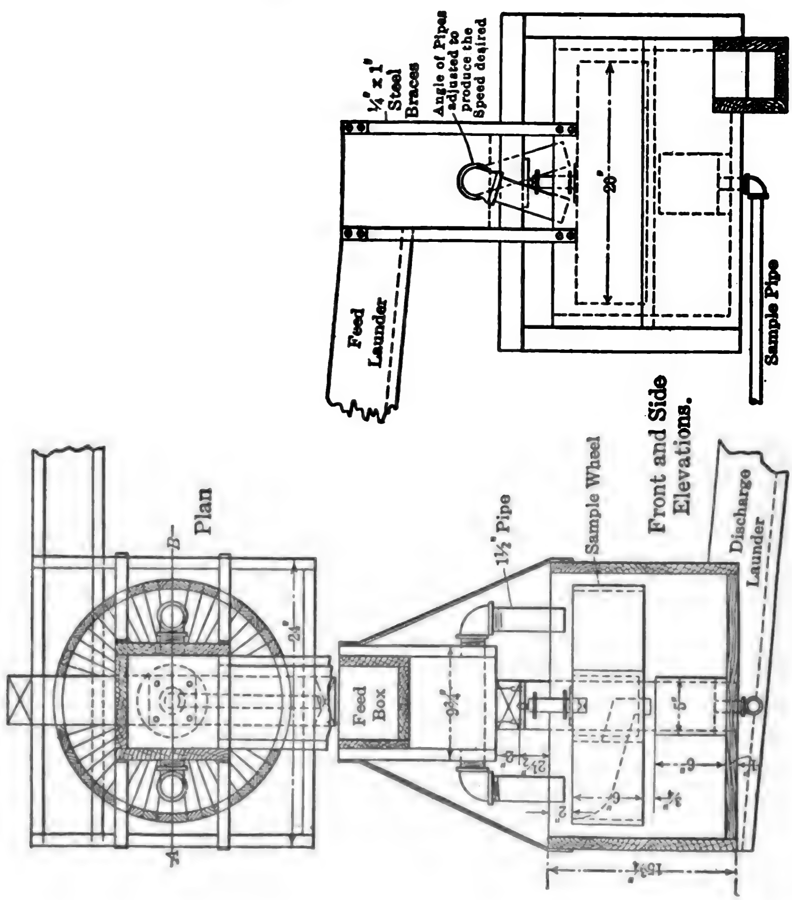


FIG. 12.—GENERAL ARRANGEMENT OF THE BORCHERDT SAMPLER.

wheel and percentage of sample cut desired. In the sampler illustrated, which is as small as advisable, 50 divisions, making a 2% cut, are used. The hub is turned in a lathe, the space marked off with dividers and the grooves made with a hack or miter-box saw. After inserting the vanes and lock strips in the rim the wheel is turned in the lathe, and the upper end of the hub faced square and counterbored to pass the free end of the bearing spindle. The lower end of the hub is also bored out as indicated, to form a drip edge around the circumference. These details are illustrated in Fig. 13.

One of the open cells is then closed off on the bottom with a wooden filler, sealed with thick paint, and a slot cut from it to the center of the hub which connects it to a closed pipe nipple screwed into the center of the lower face of the hub. A ball bearing, in this case simply the rear hub of a bicycle, is fastened to the upper face of the hub by means of wood screws passing through holes drilled and countersunk in the regular sprocket and the entire wheel is suspended by this bearing from a small iron plate bolted to the under side of the feed box, as shown in Fig. 12. From the sides of the feed box near the bottom issue pipe nipples, which carry elbows into which discharge pipes of the proper length are screwed, below the sample wheel, and so arranged that no pulp can enter it except that discharged from the sample bucket, is a small cup with closed bottom from which a pipe leads the sample to the receptacle in which it collects until removed.

As will at once be evident, the whole machine is simply a crude form of turbine, in which the pulp to be sampled furnishes the motive power, and as all the pulp is discharged through the wheel, and as all of the cells are equal and open save one, it follows inevitably that the single closed cell or bucket catches and conveys away its definite proportion of the whole pulp, regardless of the quantity of pulp (within the limit of the capacity of the sample bucket), the speed of the wheel, or the number of pipes delivering the pulp.

In use, the angle of the discharge pipes to the surface of the wheel is adjusted by turning the elbows on the thread of the horizontal nipples until the speed of rotation is as desired, generally between 30 and 60 r.p.m. This gives a large number of cuts per minute, and at the same time the wheel works without splash and requires oiling only about once a day.

In adjusting the sampler to suit given conditions as to head available and quantity of pulp to be handled, two variable factors enter, which permit of considerable latitude. Any number of discharge pipes may be used on the feed box but it is best to make the number even and to space them equally on account of maintaining the wheel in balance. The number and size of these pipes can be increased

when only a low head is available, while with the opposite conditions fewer and smaller pipes may be used, and the head in the feed box, measured from the lower ends of the discharge pipes to the bottom of the feed launder (dimension X in Fig. 12), may be increased sufficiently to flow the quantity of pulp required.

For example, the small sampler illustrated handles the tailings, including all the head water, of five tables, amounting normally to 66 gal. per min., carrying 80 lb. per min., or 2.4 tons per hr. of solids ranging between 1 mm. and slime. It has been loaded to 82 gal. per min., the limit of capacity of the Frenier pump handling these tailings, without showing any splash, and it is probable that this size would take as much as 100 gal. per min. cutting a 2% sample, as at 50 r.p.m. the same bucket would only have to handle a little less than one-quarter pint per revolution.

In this particular machine the head from the end of the two 1½-in discharge pipes to the bottom of the feed launder is 3 ft. 10 in., ample to care for 100 gal. per min. of pulp of the density stated. The rear hub of a bicycle, with its sprocket, forms an ideal bearing for the small-size sample wheel, but as the size is increased above 24 in., it is advisable to use a motorcycle hub, which, with a ½-in. spindle, provides a bearing strong enough for a 36-in. wheel. This, with a hub 12-in. in diameter and 100 blades, would easily care for 500 gal. per min., and take a 1% sample.

If it is desired to handle denser pulp than 4 or 5 : 1, or coarser than about 1 mm., the sample wheel should be modified so as to make its depth greater in proportion to its diameter in order to run the sample quickly out of the sample bucket, and in all cases the speed of the wheel should be kept moderate in order to avoid the retarding effect of centrifugal force on the discharge of the sample.

It is, of course, evident that the sample bucket could be arranged to discharge from the outer circumference of the wheel into a circular gutter or launder, but as this arrangement occupies more space and takes greater mill head, it is generally preferable to use the central discharge.

Frequent tests with measured weights have shown that this sampler, under all operating conditions, cuts the definite percentage intended, and a convenient means is thus provided for measuring the flow of pulp in a launder, or figuring the tonnage handled by the fines department, since the weight of sample collected multiplied by the percentage factor of the sampler gives the total tonnage passed.

A convenient means of collecting the sample consists in piping it from the sampler to one of two oil barrels stood on either side of the pipe which has an elbow even with the center of the top of the barrels, from which leads a piece of pipe long enough to reach across and dip below the water

level in one barrel. The barrels are used alternate days and are put in service merely by swinging the end piece of pipe on its elbow so as to discharge into the other barrel. The overflow from the barrels is clear, and the collected sample is prepared for the drying oven by stirring up the settlings with that part of the supernatant water which cannot be dipped or siphoned off clear, and pouring the mixture through a riffle sampler until a sample of the desired size results.

Sampler for Lead Concentrates.—The sampler shown in Fig. 14 was designed and used by a southeast Missouri lead company for sampling carloads of concentrates. The sampler cuts a core from the top to the bottom of the car and retains the wet sludgy lead at the bottom, thus giving a sample which represents the true moisture and lead percent-

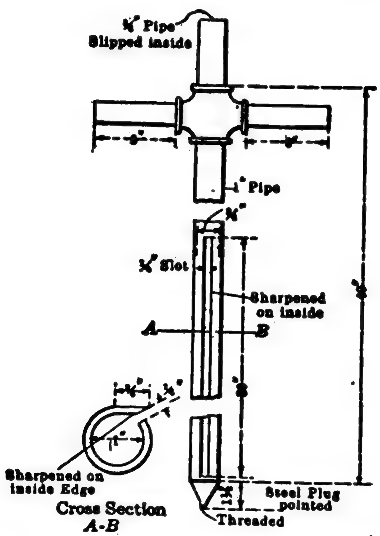


FIG. 14.—SAMPLER FOR LEAD CONCENTRATES.

age of the shipment. The ¾-in. pipe slips inside the outer one and the sampler is thrust to the bottom of the car, either by shoving or hammering the top of this pipe which extends above the sampler proper. The inner pipe is then removed and a few turns given the sampler which causes the sharpened slot to cut an even sample. The sampler is then pulled up and the lead knocked out through the top onto an iron pan. The ¼-in. slot in the sampler is made by prying open one edge along a cut. The inside edge of the projecting side of the slot is filed down to a sharp edge.

In sampling a car it is first marked off into 2-ft. squares in each of which a sample is taken, this giving from 48 to 64 holes to the car. With vanner concentrates there is sometimes a suction which will pull out part of the sample. This may be avoided by loosening the pipe in

the hole as much as possible and then removing it with a twisting motion. The samples are emptied on a sheet of iron and cut down by quartering in the usual manner. This method of sampling has proved satisfactory, checking within less than 1% with the smelter's samples which are taken every twentieth shovelful.

A Pilot Concentrator.—An interesting and useful idea for concentrating mills has been put into operation at the mill of the Coniagas Mines Co., Ltd., at Cobalt. It consists of designating one concentrator as a pilot, which gives an accurate idea of how the mill in general is operating. A sample is cut automatically from the tailings as they go to the dump and this sample is sent back to the pilot table where it is reconcentrated. If the mill is not doing the best possible work a further recovery is shown on this table, while if all the tables are doing good

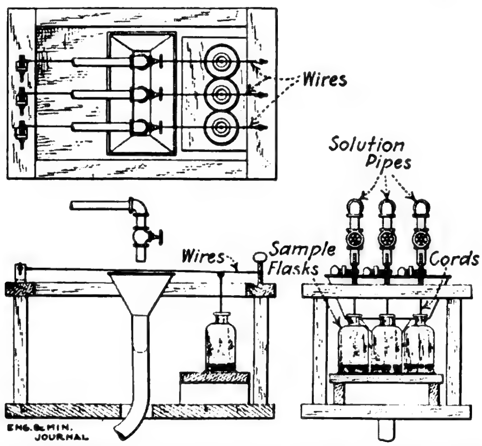


FIG. 15.—SIMPLE DEVICE FOR SAMPLING SOLUTIONS.

work the pilot will not show any additional recovery. Thus a glance at it will show at any time just how the mill is being operated. The idea originated with Frazer Reed, mill superintendent for the Coniagas company.

CYANIDE SOLUTIONS

Wire Sampler for Cyanide Solution (By H. P. Flint).—An ingenious and cheap device has been in use in an Arizona cyanide plant for automatically sampling solutions. The general arrangement is shown in Fig. 15. Three grades of solution are brought to the sampler in ½-in. pipes, and the stream from each pipe is cut by a piece of baling wire attached to thumb-screws at each end for adjusting its horizontal position in the stream and its inclination. A small quantity of solution flows down the wire and is intercepted by a knot and directed down a cord into

a bottle in which the sample is collected. The cord is weighted at its lower end. The reject is caught in a hopper and conducted to a lower tank. The pipes have valves set in the vertical nipples to control the flow of solution.

Device for Sampling Zinc-box Solutions (By Fred. W. Monahan). —A simple and useful method of sampling the solution in the last zinc box in cyanide mills consists in the use of a wick which continuously siphons a small amount of the solution over into a bottle, thus taking a uniform and accurate sample. A cotton wick is used with one end in the zinc-box solution and the other in the bottle to receive the sample. The wick passes through a porcelain insulator on the edge of the box, and

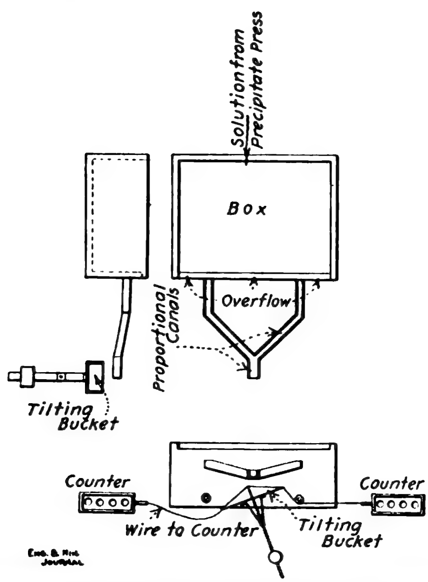


FIG. 16.—ARRANGEMENT FOR SAMPLING AND WEIGHING SOLUTION AT HOLLINGER MILL.

has another insulator attached to the end to act as a weight. The bottle is set below the level of the water in the zinc box. The samples are collected each day and tested for any trace of gold which may have passed the zinc shavings. This is a much more accurate method than to have the boy who bucks the samples go around and take out a dipperful now and then. It is also a means of checking and often preventing large losses due to careless treatment of the zinc shavings.

Solution Sampler and Weigher.—At the mill of the Hollinger Gold Mines, Ltd., Porcupine district, Ont., the cyanide solution from the Merrill precipitation presses is passed through a device known as a Black Hills sampler and weigher. This device, illustrated in Fig. 16, consists of a box, one side of which is slightly lower than the other three, and set

exactly level, so that the flow of solution over it may be equal at all points. In front of this overflowing side are set two small canals which take out a determined proportion of the entire flow, and deliver it to a tripping bucket. When this bucket is filled to a certain point the weight of the solution dumps it and at the same time presents the opposite bucket to be filled. A counter is arranged to count the number of times the bucket has tripped and, knowing the weight of each bucket of solution and its relation to the total flow, the total quantity of solution which has passed through the device can be calculated. It is calibrated from time to time, but the data secured are practically constant. While the buckets are discharging, a part of their content is caught in a bottle, this serving as a sample of the precipitated solution.

COPPER BULLION

Saw Sampler for Copper Bars.—Copper bars are sampled at Cananea with a series of six parallel saws held in a framework. The bar is inserted and cut halfway through. An electric attachment rings a bell notifying the attendant that the saw should be stopped. The bar is then turned over and cut on the other side midway between the first cuts. The device saves labor and gives a more accurate sample than the rip saw that was formerly used.

Top and Bottom Drilling in Pig Copper (By Donald M. Liddell).—It seems to be an established fact in the sampling of copper bars that samples taken by drilling from the top of the bars will not check with those obtained from the bottom. There is no fixed rule in the case, that is, all samples drilled from the bottom are not richer than those drilled from the top, although the majority of pimple-copper samples seem to be, so that one might be led into thinking that the difference was simply due to a different order of drill holes with regard to the pigs, *i.e.*, that if the pigs were numbered 1, 2, 3, etc., and the templet holes correspond to this order on the first drilling, that in drilling again, hole No. 3 might fall in pig No. 1, etc., producing a small assay variation. However, if this were the case, the matter would probably even itself up in the course of drilling several lots, whereas the richness of top over bottom or bottom over top remains fairly consistent in any given brand of copper. Table I shows the differences springing from this source.

The differences appear to arise chiefly from the following causes. When the drill strikes the copper it produces fine dust for the first few revolutions. This dust is, as a rule, much richer than the average of the drill hole, and being caught by the unevenness of the surface in pimple metal is not brushed into the sample, so that the sample taken by drilling from the top will be too low in value. The bottom of the pig is smoother

and does not hold the dust from its surface to as great an extent, nor does blister copper. Moreover, there is probably a certain amount of dirt brushed into the sample from the top of the pig, which lowers the value. In drilling from the bottom the drill will probably break through the last of the hole carrying away chunks from the surrounding surface. These are richer than the average of the rest of the sample and consequently raise its value. In drilling blister copper from the top the drillings are likely to be thrown into the blister around the drill and escape getting into the sample, thereby making it poorer. This can be prevented to a great extent by smashing the blisters down with a heavy hammer before beginning the drilling.

Table I.—Differences in Top and Bottom Sampling

	A	B	C	D	E	F
Copper, %						
Drilled from top..........	99.085	98.986	99.068	98.872	98.356	99.027
Drilled from bottom......	98.977	99.058	98.896	98.805	98.290	99.095
Silver Oz.						
Drilled from top..........	36.70	50.49	71.40	96.02	240.01	35.55
Drilled from bottom......	37.54	50.40	73.72	95.815	238.85	34.05
Gold Oz.						
Drilled from top..........	13.468	0.874	0.496	15.953	7.677	1.296
Drilled from bottom......	13.539	0.875	0.514	15.883	7.632	1.281

Each of the above represents an average of 5 to 20 lots. A, B and C, pimple finish. D, E and F, blister finish.

Table II.—Assay of Skin of Copper Pig

Depth	Ag, oz.	Au, oz.
Surface to 1/8 in.	146.2	2.58
1/8 in. to 1/4 in.	129.2	2.54
1/4 in. to 3/8 in.	135.2	2.56
3/8 in. to 1/2 in.	134.2	2.55
1/2 in. to 5/8 in.	134.6	2.56
5/8 in. to 3/4 in.	131.5	2.52
Top burs	160.0	2.35
Bottom burs	100.1	1.44
Assay of lot	91.7	2.125

Concerning the rich "skin" of a copper pig, Table II of assays may be of interest, the samples being taken by removing one layer after another of a pig. The last three results were obtained by picking out all burs obtained from the top and all burs from the bottom of a complete lot. It seems needless further to multiply examples, the general conclusion

being that in copper bars there exists a thin skin on both the top and bottom of the pig, which is very much richer than the inside, and that any sampling which does not allow for this fact will be more or less incorrect. It seems as far as my experiments go, that in pimple-copper, samples obtained by drilling from the top will be richer than those obtained by drilling from the bottom, while the reverse seems true in blister. From furnace runs on weighed material it also seems to be established that neither sample is correct, but that the average results obtained by drilling one-half from the top and one-half from the bottom will be close to the truth. With anodes (furnace-refined copper) there seems to be little difference between samples obtained by top and by bottom drilling, yet even here it is probably best to drill half from the top and half from the bottom.

Influence of Number of Templet Holes in Sampling Copper (By Donald M. Liddell).—It appears that in the sampling of pig copper by the ordinary templet drilling, as the number of holes in the templet is increased, the silver assay diminishes up to a certain point, after which it remains fairly constant. Table III is given herewith showing the influence of the number of holes in the templet on the silver assay, the number of holes being those per quarter bar.

It seems that the explanation of this may lie in the following: If the surface of a copper slab be examined at about the same distance from the edge that the pig is thick, there will be found a well-marked line, showing where the cooling of the slab from the bottom upward has met the cooling of the pig from the upper outer edge inward. The drillings from

TABLE III.—EFFECT OF TEMPLET HOLES ON SILVER ASSAY

Holes	Silver assay	Holes	Silver assay
2 × 3	76.1 oz.	5 × 8	73.8 oz.
3 × 4	75.7 oz.	7 × 11	73.7 oz.
4 × 5	74.8 oz.	8 × 12	73.8 oz.
4 × 7	74.7 oz.	....	....

TABLE IV.—SEGREGATION OF SILVER IN COPPER

Hole numbers	Brand A oz. per ton	Brand B-oz. per ton	Brand C-oz. per ton
1–13	133.69	62.6	404.9
14–24	149.94	69.0	454.7
25–33	136.65	63.41	443.1
34–40	145.05	63.81	....
41–45	151.00	63.10	....
46–48	150.09	60.60	448.02

along this line are usually much richer than those from either side of it, and are sometimes richer than any others in the slab. Typical assays, showing this enriched zone, are given in Table IV, accompanying Fig. 17, the tests shown having been made on a quarter-section of a slab $18\frac{1}{4} \times 30 \times 1\frac{1}{2}$ in., weighing approximately 225 pounds.

If one consider the case of a copper bar drilled as shown by the X's on a 4×5 templet and as shown by the O's on a 6×5 templet, Fig. 18, it will be seen that the two vertical rows of drill holes *A A* will, in each case, just about hit the enriched zone, while in the case of the 6×5 drilling there will be two rows of holes in what is probably a poorer portion

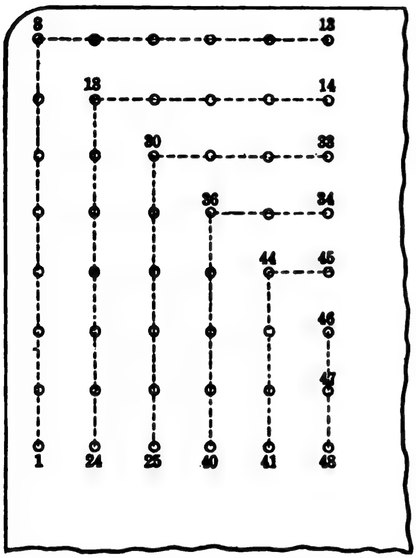


FIG. 17.—TEST DRILLING ON COPPER BARS.

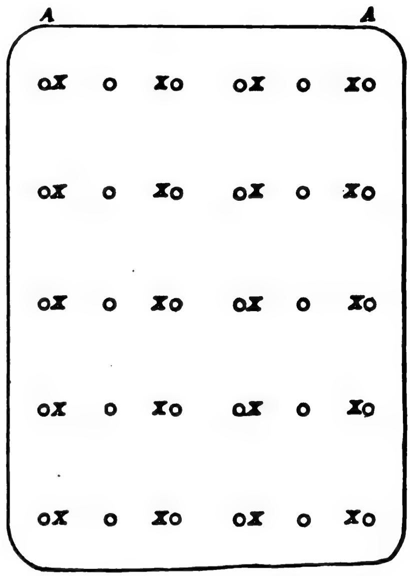


FIG. 18.—TEMPLET-HOLE DIAGRAM.

of the pig, to decrease the general average. If the number of vertical rows (as they come in the illustration) is held a constant, and the number of horizontal rows increased, the same argument holds, while in actual practice we get the twofold benefit of increasing the number both ways simultaneously.

What the gold does is not discussed above, since in general the gold variations are much less widely marked than the silver, and the probable error of observation bears a higher ratio to the contents. As an interesting addendum to the above assay tables may be noted one of reversed segregation due to lead in the copper, where the outside row of holes showed 66.7 oz. Ag, the extreme corner hole 67.9 oz. and the inside hole 65.7 ounces.

Moisture in Copper Bullion (By Donald M. Liddell).—Although moisture in copper bullion is not a common occurrence, it is occasionally present; and if not tested and allowed for, is as great a source of loss as unobserved water in ores would be. The following is a record of experiments on that brand of copper on which the first long-extended set of moisture tests was made, as far as my own knowledge goes. The reason for these tests was found in a long series of careful furnace runs in which strict account was kept of all copper bullion charged into and taken out of the furnace, which showed a constant loss that could not be explained reasonably on any hypothesis of stack losses.

Moisture, of course, seemed to be the only explanation of such a phenomenon, although at that time only surface moistening was suspected. To ascertain the capacity of the bullion for such moisture, six rain-soaked pigs weighing 1406 lb. were dried at 200° F., losing only 5.5 lb. in the entire weight. At the same time five apparently dry pigs were taken from the yard and dried at 200°, but these suffered no ascertainable loss. Still following up these experiments 50 pigs were taken and dried in square iron pans at about 200 to 212°. On reweighing them 6 lb. loss was observed, but this was inconclusive, as they had been handled so often and so violently as to dislodge a large amount of dirt and scale. However, when the samples were being drilled a few days later it was noticed that the drillings seemed perceptibly moist. Three of these pigs were leaned against one of the furnaces where they became so warm as to be uncomfortable to the touch. They were reweighed while still warm 24 hr. later, but showed no perceptible loss, although the previously taken drillings from the same lot when dried showed a loss of ½ oz. on 104½ oz. This led to the belief that the moisture in these pigs was so occluded as to make it impossible to drive out by ordinary heating.

Following up this the drillings from 750 pigs were tested, dividing them into 15 lots of 50 each. This test on about 100 oz. of drillings per lot, showed moisture from nothing to 1.39%. Another test was then run on 50 pigs which were covered with canvas as quickly as taken from the ship on which they were received, drying them as before in the iron pans at about 200 to 220°, but the loss for the 50 was only 6 lb. in 11,712. Not discouraged by the above list of negative results, a more determined attempt was made to determine the moisture on the pig itself and for this purpose 885 pigs were taken from a shipment of 800 tons piled in the yard of the works and dried for some hours at temperatures ranging from 200° to upward of 500° F., every pig of the 885 reaching 300° or over at some period of the drying. The weighings and dryings were as far as possible made in batches of 32, weighing about 7160 lb. These pigs were afterward drilled and the drillings tested for moisture by weighing before and

after drying in a steam bath. Table V shows the result of these tests as well as a second set run on 700 additional pigs.

TABLE V.—PIGS DRIED IN PANS

	Wet weights	Dry weights	Loss	Per cent., loss
Pigs, pounds	196,720.5	196,035.0	685.5	0.348
Drillings from dry pigs, grams	41,683.53	41,675.43	8.10	0.019
Drillings from original undried pigs, grams	173,930.0	173,424.0	506.0	0.291
Pigs, pounds	149,790.0	149,202.0	588.0	0.393
Drillings from dry pigs, grams	30,927.64	30,922.90	4.74	0.015
Drillings from original undried pigs, Troy oz	3,240.10	3,231.39	8.71	0.269

Another series of experiments was run, heating the pigs to about 400° F. in a furnace used for annealing sheet copper. The pigs were left in five or six hours or longer, and were thoroughly warmed. These results are shown in Table VI. Drillings taken from the 550 dried pigs in Table VI, lost only 6.26 grm. on 27,379.75 or only 0.023%. The weighted average of the above results on direct moisture determinations in the pig (not those on the drillings) is 0.425%. In the original experiments the apparent furnace losses were 23,835 lb. of copper on 4.463,-319 lb. of bullion, or 5.35%. If we allow that 0.425% of the bullion was water the loss sinks to 3982 lb. of copper, which probably indicates that the moisture as found is a trifle low.

TABLE VI.—ANNEALING FURNACE DRYING

Number pigs	Net weight, pounds	Dry weight, pounds	Loss, pounds	Per cent., loss.
550	121,710	121,207	503	0.413
100	22,640	22,553	87	0.387
350	78,775	78,376	399	0.507
350	78,605	78,186	419	0.533
600	134,864	134,156	708	0.525
100	22,498	22,409	89	0.396
500	110,817	110,359	458	0.413
200	44,518	44,378	140	0.315

That the moisture spoken of is not surface moisture, due to rain, etc., is shown by three things. (1) A temperature of about 250° F. is necessary to evaporate this moisture; (2) a carload of the copper soaked with a stream from a fire hose and allowed to dry spontaneously in air, came back to its original weight; (3) last and most indisputable, when the consignors wrote that they would adopt a method of cooling their pigs

which would render them moistureless, the moisture tests on the pigs dropped to 0.033% on a lot of about 88,500 pounds.

It may be added as an interesting fact, that the moisture tests on the pig-copper drillings while they led to the truth, were in themselves unreliable. It would occur to any one that they would probably be a trifle low owing to the heating of the drillings by drill friction, and consequent evaporation of the moisture of the drillings. Apart from this there is another factor, *i.e.*, that copper apparently acts in much the same manner that platinum does, in attracting and condensing a film of air and moisture on its surface. The investigation of these peculiarities of copper drillings from furnace-dried copper showed minus moisture, *i.e.*, the drillings showed an increase in weight on drying. While ascribing this to oxidation, a more thorough test was made by taking bars of furnace-refined copper, free from any blow-holes or cavities and drilling them, cleaning the surface beforehand carefully. The first two tests on 482.28 grm. showed losses of 0.20 grm. and 0.16 grm. respectively when the dried drillings were weighed hot, even though the drillings were oxidized. Three further tests were tried, as shown in Table VII. Sample No. 3 was discarded and the other two sent back to the drybox for 16 hr. at 270°; they weighed hot, 483.14 and 483.20 grm. After cooling in air, the day being fair and warm, they weighed 483.24 and 483.26 grams.

TABLE VII.—MOISTURE TESTS ON COPPER DRILLINGS

	No. 1 Dried at 200° F.	No. 2 Dried at 200° F.	No. 3 Dried at 270° F.
Weight before drying, grams	483.07	483.12	110.24
Weight after drying, grams	483.02	483.08	110.25
Loss, grams	0.05	0.04	......
Gain, grams	......	......	0.01
Remarks	Unoxidized; All drillings weighed hot	Unoxidized	Oxidized

Since these first tests were run other brands of bullion have been tested, the general deductions being as follows: That any bosh-cooled pig is likely to contain moisture; that such moisture is occluded in such a manner as to render it difficult if not impossible to drive off under about 240° F.; and that while the moisture tests should be run on the pigs themselves at or above that temperature, moisture tests on the drillings answer fairly well as a preliminary measure.

Magnetic Particles in Copper-bullion Sampling (By Donald M. Liddell).—The occurrence of magnetic particles in copper-bullion samples

is a source of some perplexity to the assayer, the question being whether or not to remove them before assaying. While there can be no doubt of the propriety of so doing in the case of wirebar, cathode and well-refined anode samples, it is questionable whether they should be taken out of converter bar drillings.

This matter has been tested in two ways, first by the direct method of weighing the drills and the grinding machinery on silver-bullion scales and noting the loss sustained on treating a known weight of copper; second, by removing the magnetic particles from bullion samples, and assaying the portion removed. In order to reduce the experimental error incident to the first method, the same drills and grinder parts were used again and again on consecutive experiments, so that the error of one experiment was absorbed by the next. The contamination of the sample occurred in three stages; drilling; rough grinding; fine grinding—in the drug mills made by Hance Brothers & White.

TABLE VIII.—IRON INTRODUCED BY GRINDING MILLS

Coarse grinding %	Fine grinding %	Total %
0.0014	0.0165	0.0179
0.0048	0.0122	0.0170
0.0048	0.0098	0.0146
0.0035	0.0159	0.0194
0.0024	0.0130	0.0154

The amount of steel lost by the drills amounted to the negligible quantity of 0.0043% of the weight of copper drillings made. In the grinding mills the results for five consecutive weeks are shown in Table VIII. The introduction of iron particles was greatest when the mills were new. About 900 lb. of copper was ground in the coarse grinding and about 200 in the fine grinding in each week. It was, of course, impossible to run the drills continuously for this long a period, owing to their needing regrinding. The results as to the loss by the drills cover two days only.

TABLE IX.—ANALYSES OF BULLION AND MAGNETIC PARTICLES

	Bullion			Magnetic Particles			
	Cu, %	Ag, Oz.	Au, Oz.	Cu, %	Ag, Oz.	Au, Oz.	Fe, %
Brand A	99.050	75.65	14.035	46.95	47.3	4.30	.....
Brand B	97.825	352.20	8.375	34.00	143.8	13.60	.....
Brand C	99.000	......	......	36.40	.....	.....	31.28

On the direct analysis of the magnetic particles themselves, the results shown in Table IX indicated that to remove all of the magnetic

particles from copper bullion is an error, as they consist chiefly of converter slag. The nearest approach to theoretical perfection would be to remove all particles just after the coarse grinding, the introduction of foreign iron up to that time being less than 0.01 %; perform the fine grinding, again remove all magnetic particles and throw them away, then replace the particles removed after the first grinding.

A Short Formula for Samples Containing Metallics (By Donald M. Liddell).—It is necessary occasionally to sample material containing large quantities of metallics, such as reverberatory slags, where with successive crushings the metallic portion must be thrown out. If it be assumed that the metallics are homogeneous, the accompanying formula may be useful.

Let the weights of metallics taken out at successive crushings be represented by A, B, and C, and the corresponding weights of material not metallics by D, E and F; then the total percentage of metallics is:

$$100\left(1 - \frac{D}{A+D} \times \frac{E}{B+E} \times \frac{F}{C+F}\right).$$

If C and F be the final metallics and pulp, and their separate assays are c and f then the assay of the entire original pile is:

$$c + (f - c)\left(\frac{D}{A+D} \times \frac{E}{B+E} \times \frac{F}{C+F}\right)$$

It will be noted that while A and D, B and E, and C and F must be similiar units, it is by no means necessary that D, E or F be the same units, one may be pounds, another ounces and the third grams.

II

ORE DRESSING—BREAKING, CRUSHING AND GRINDING

GENERAL REMARKS

Notes on the Construction and Operation of Stamp Mills[1] (By G. H. Fison).—Although the power consumption is most economical when the weight of stamp is as great and the drop as low as practice will allow, the duty per stamp controls the operation of a mill, this being adjusted to the capacity of the mill to reduce the ore to the required fineness and expel the pulp through the screen. To maintain a high stamp duty, the speed and height of drop of stamps must be arranged to give the largest capacity of screen discharge.

At various times efforts have been made to increase the screen discharge by increasing the screen area. Mortar boxes with double discharge have been tried, but usually have not found favor with mill men. One cause for this is that the rear screen is more liable to damage and the consequent loss of time in renewal counterbalances the extra discharge gained. The tables also have to be lowered into a less conveniene position in relation to the discharge of the mortar in order to receive the flow from the back of the box. However, if these matters are cone sidered in the original designing of the mill, such difficulties should be easily overcome. In using a back screen a high discharge, 6 in., would be necessary to avoid banking against the bottom edge of the screen under the feed chute.

The life of all stamp mills is materially affected by the vibrations to which they are subjected. These may be minimized by framing timbers carefully and using cushions of gasket rubber under mortar boxes, king-posts and countershaft boxes. The use of concrete mortar-box foundations is another important factor in reducing vibrations in the mill. It is also advisable to carry countershaft bearings on separate concrete blocks and to have the ore-bin foundations built independently of the mill framing. Maintaining the king-post bolts in good order with the nuts tight is also of prime importance. It is therefore evident that these bolts should be placed so as to be accessible and in plain view so that they can be watched continually. It is well to use a heavy yoke with the bolts so as to give a clear height above the feeder platform and thus

[1] Abstract of an article in *Min. Journ.*, Sept. 4, 1909.

make the nuts accessible for heavy spanners. The bolts holding down the mortar box are also important factors in the life of a stamp mill. They should never be placed so as to be masked by mill-water mains or other avoidable obstructions so that they cannot be conveniently reached with heavy wrenches. It is well to have these bolts made in two lengths and joined in the middle with hooked ends. This enables a defective screwed end to be replaced by unhooking the upper half of the bolt without having to dig into the foundation, as is the case when a solid bolt must be removed. Self-locking nuts are reported to be successfully used in a number of mills.

The present tendency seems to be to use boxes as narrow as possible with corners rounded in order to avoid waste spaces. With the narrow boxes the inside depth varies from 4 to 9 in., measuring down from the screen opening. The advantage of using a shallow box is that it allows the height of the discharge to be regulated by chuck blocks, while in deep boxes wear has to be compensated by packing up the dies. Mortar-box liners are made of hard cast iron, mild cast steel or chrome or manganese steel; they should be interlocking and self supporting. They then require but one hard-wood wedge to be driven down in either end of the box to hold them tight. A set of interlocking liners can be renewed within 10 min. after the time the dies are removed from the box. Liners should not extend below the level of the false bottom.

Except where inside amalgamation is used the height of the discharge should be just as short as is consistent with avoiding the banking of sand against the lower edge of the screen, say from 1½ to 2½ in. above the top of the dies, according to the depth of feed employed. The dies should be turned about once in two weeks in order to equalize wear, the highest edge being placed at the back of the box. They should also be proportioned so as to wear out simultaneously. A 7-in. die will usually last as long as a 12-in. shoe. This may in some cases be regulated by using different materials for the die and shoe.

The best results are usually obtained with speeds varying from 95 to 105 drops per min., with corresponding falls of from 8½ to 6 in., varying of course with the type of mill and quality of the ore crushed. In this connection it should be remembered that the effective drop of a stamp is from ½ to 1½ in. less than the height raised by the cam.

Hardwood guides are reliable and when used with proper care will last for years. In some sections, notably Rhodesia, a grooved cast-iron guide made in halves is coming into favor. The bore is provided with annular grooves to form receptacles for a lubricant consisting of four parts of soft soap to one of graphite. Work is facilitated, when taking up the wear in the guide blocks, by having permanent marks on the king-posts and guide beams to show the exact stem centers. It is

very important that these be maintained especially with narrow mortar boxes.

Plain copper plates are preferable to silver-plated ones in that when once set with gold amalgam they will hold a thicker layer of mercury than will the silver-plated ones. It is, however, more difficult to start the amalgamation with the copper plates. When using silver-plated amalgam plates, great care must be exercised in steaming and in scraping or the silver plating will be removed with the amalgam. Hard scraping at any time is liable to injure the plated surface. With acid mill water lime should be fed to neutralize the scouring effect. Silver amalgam is sometimes used with success as a first dressing on the bare copper plates, thus enabling a good gold saving to be made from the start and producing a surface more absorbent and sensitive than is that of the silver-plated copper. Cyanide solutions should not be used in dressing plates unless the greatest care is exercised. Although the plates will be kept bright, gold losses will result and the plates will not be kept in that solid and sticky condition induced by proper scouring and rubbing. Increased absorbtion of mercury by the gold is caused by raising the temperuatre of the feed water; when it can be maintained without variation at about 80° F., good results are obtained.

The water to be supplied a mill is governed more by the flow over the plates than by the conditions in the mortar box. The finer the rock is crushed the more water will be required to keep the plates free from sediment; an excess of water will tend to carry black sand and fine gold away. The best way of introducing water into the mortar is to direct the stream so that it strikes low down the back of the box, and washes over the die with sufficient force to carry fine particles through the screen, but with insufficient force to carry the uncrushed ore from the die. Water with a constant head should be supplied in all cases, and the branch leading to each battery should be fitted with two cocks or valves, one to be used as a regulator and to remain set to pass the required amount of water; the other to be used, whenever the mill is stopped or started, for opening and closing the total supply to the box. A dial may be used with advantage on the first cock so that the exact opening can be observed and maintained.

No generalization can be made regarding the loss of mercury in amalgamation. Under ordinary conditions, however, with a 7-dwt. plate recovery, the loss, including the process of retorting, should not exceed 1 dwt. of mercury per ton of ore crushed. Where inside amalgamation is practised the losses in many cases are increased. The use of good traps in the tailings launders aids materially in reducing the quicksilver losses. In special cases the use of blankets is of undoubted advantage and economy. It is a good scheme to have a blanket mounted on rollers

as an endless belt revolving slowly against the stream of pulp; the under side of the blanket passing through a trough of water and being washed automatically, thus saving the dirty and disagreeable work of constant hand washing.

A 10-Stamp Mill of Novel Design (By J. Bowie Wilson).—The 10-stamp mill shown in sectional elevation in Fig. 19 was erected at Sandstone, Western Australia, a town in the East Murchison goldfield.

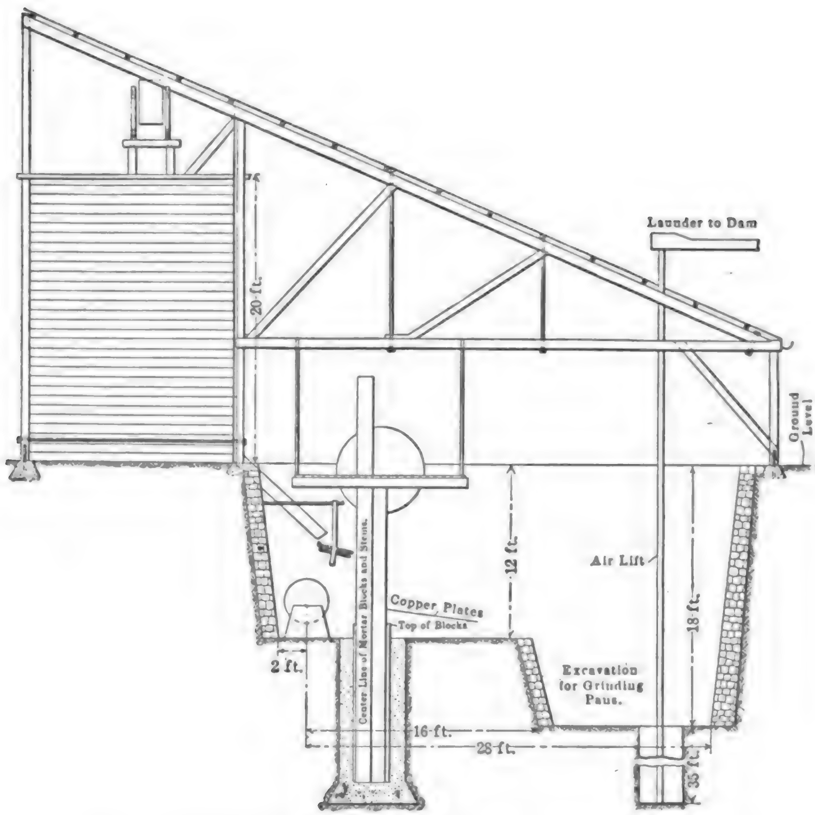


FIG. 19.—TEN-STAMP MILL AT SANDSTONE, WESTERN AUSTRALIA.

At the time the plant was being erected the railway was 100 miles from Sandstone so all material had to be brought in by wagon. The conditions prevailing at Sandstone, and throughout most of the Western Australian goldfields, necessitate the erection of plants upon practically a level plain where there is no local timber suitable for building purposes. There is so little fall to the ground that it is necessary to lift tailings to the cyanide vats. When considering the general layout of the plant the idea of excavating the battery site and placing the mortar boxes below

the level of the ground was suggested. After carefully going into details of costs this plan was adopted and the plant was erected and now has been running for over a year, giving complete satisfaction.

The mine had three shafts up any of which it might be advisable or necessary to raise ore. For this reason the rock breaker was placed close to the bins of the most important shaft and with the top of the breaker level with the ground. This necessitated the excavation of a pit for the rock breaker in order that any ore raised from either of the other shafts could be trucked along the surface and tipped directly to the breaker. If the breaker had been erected in the head-frame of the main shaft or upon the surface of the ground it would have been necessary to make arrangements to elevate the ore. The only disadvantage of this design was that by lowering the discharge of the breaker the height to which the broken ore had to be elevated to the battery bins was increased. The broken ore from the breaker fell upon a belt conveyor which elevated it to the battery bin which was built at such a distance from the shaft as to allow the belt to rise to the required height. The battery bin consists practically of a bottomless timber box placed directly upon the ground. It was built of timber posts, the bottom set in concrete and the tops braced across either with tie bolts or old hoisting rope. Tie bolts were also put in at the base. Lining boards of 2-in. thickness were used on the sides. No bottom was placed in the bin but low-grade ore was first dumped in direct on the leveled surface to form a rock bottom at the ordinary angle of repose of the broken ore. The delivery end of the belt conveyor, elevating the ore from the rock breaker, was carried at such a height above the middle of the bin that the broken ore heaped up in the middle of the bin and sloped away to the sides. This materially increased the capacity of the bin without adding to the cost of erection. The building of the bin in this way, resting directly upon the ground, reduced the timber required for construction to a minimum, which was an item worth considering when all the timber had to come 200 miles by rail and 100 miles by road from the nearest port with no back loading to reduce freight. It also reduced the length of conveyor belt required as it reduced the height to which the broken ore had to be elevated. The box form of bin, of course, necessitates a certain amount of ore remaining dead in the bin but then this can be looked upon as a reserve of ore on the surface to be used in case of accident in the mine to keep the plant running and even if it does require extra handling to get it into the battery, any manager running a small mine will appreciate that such a reserve of ore on the surface is an advantage.

The broken ore passed by gravity in the usual way into automatic ore feeders to the mortar boxes which rested upon the top of the mortar blocks, that were 11 ft. below the level of the ground. As will be seen

in Fig. 19, below the copper plates the excavation was deepened another 6 ft. to allow for the installation of pans as the plant was designed to treat a large tonnage by coarse crushing in the mortar boxes followed by fine grinding in pans. Below the pans the pulp flowed to a sand lift which elevated it to the launder running to the settling dam.

In making the excavation to contain the mortar blocks and grinding pans a ramp was cut, up which tip drays could be hauled. This was afterward used as a belt way for the drive from the battery engine which was placed upon the ground level to the rear and to one side of the ore bin. The material excavated could be used to slightly bank up the surface around the excavation to prevent it being flooded during rains and the remainder was useful in making dams to impound water and tailings as the whole of the water required for the plant had to come out of the mine and was reused after settling in dams.

As will be noticed from the plans this method of construction allows of a minimum of timber in the whole construction and it also makes a compact plant when erected. It further has the advantage that there is only one way into the battery and it is harder for anyone to pay a surreptitious visit to the plates for the purpose of seeing how the amalgam is getting on. In places where timber is scarce and expensive but the surface is easy to excavate, this design is worthy of consideration.

The chief drawbacks that can be advanced against this design, are the liability to flooding and the cost of raising the pulp an additional 18 ft. The first is more evident than real and if occasionally the excavation does become flooded there is nothing below the water that would be damaged as the driving machinery is not below ground. The second condition must depend upon individual circumstances when the cost of elevating the tailings the additional distance must be placed against the savings in the cost of erection. With many small plants it is frequently sound to knowingly slightly increase working expenses in order to reduce the original cost of erection.

The power to drive the battery and an air compressor for the mine was supplied by two National producer-gas engines. They were run upon "mulga" charcoal, mulga being a low growing scrub which covers the district. Producer-gas engines were installed to reduce the firewood bill, as mulga big enough for use in boilers was becoming scarce and had to be hauled long distances. Also all the water available came from the mine and carried a high percentage of solids which caused great trouble in the boilers and only allowed the use of the most simple and uneconomical types and even these had to receive a great deal of attention and repairs. The producer plant was erected and started by engineers who had never run this class of machinery but from the start it gave little trouble and has been highly successful.

Crushing and Classification at Homestake.—In discussing the paper of A. J. Clark and W. J. Sharwood on "The Metallurgy of the Homestake Ore," read before the Institution of Mining and Metallurgy, W. A. Caldecott takes up (*Bull.* 101, I. M. M.) certain considerations affecting the Homestake crushing and classification practice, as compared with the methods followed on the Rand, and which present a certain analogy as involving the treatment of large tonnages of low-grade ore in which a high percentage of the gold content is readily recoverable by amalgamation, followed by cyanide treatment; concentration and roasting being unnecessary in either case. The main characteristic of the Homestake equipment appears to be the large number of small units involved in handling a large daily tonnage. The thousand 900-lb. stamps, and the large number of cone classifiers employed, are striking illustrations of this tendency, which is counter to the policy in vogue elsewhere of utilizing as large units as possible for operations on a large scale.

The Homestake ore appears soft, which renders comparisons of crushing capacity with hard pyritic banket ore difficult, and the grading analyses given are based on screens with apertures different from those in common use on the Rand. The cost of stamp milling and regrinding 2795 tons daily at Lead may be calculated from data in the paper to be 1s. 6.04d. per ton, and the pulp leaving the crushing plant to contain 19.3% + 100 mesh (0.0057 in.), and 80.7% − 100 mesh. The cost of crushing 2542 tons per 24 working hours at the Simmer & Jack Proprietary Mines, Ltd., during October, 1912, using a combination of steam-driven gravity stamps and of tube mills with more costly electric drive, was 1s. 4.453d. per ton, apart from water (0.500d.) and pulp elevation (0.819d.) costs, and the pulp leaving the crushing plant contained 28.8% + 90 mesh (0.006 in.) and 71.2% − 90 mesh.

The ratio of water to ore in the Homestake battery pulp is stated as 11 : 1, as compared with 4.5 : 1 in the pulp leaving the crushing plant at Simmer & Jack. The high discharge (10 in.) of the Homestake batteries necessitates the use of large volumes of water to wash out the crushed ore, but, even with cheap water and lime, the doubled volume of pulp produced entails a corresponding increase of amalgamated plate area and of classifiers for dealing with the pulp later, and may thus partly account for the 101 cones employed at the No. 1 plant, as against the fraction of this number which would be needed for larger units with a more concentrated pulp.

The stamp duty of 4.2 tons, based on the authors' data, is, of course, due to the use of light stamps and fine battery screen with high discharge. Although tube mills have been employed on a limited scale at the Homestake, the fact that crushing banket ore by their means costs 20% less than by stamps would appear to justify the consideration, even apart

from the installation of heavier stamps, of shutting down at least half the thousand stamps, and replacing them by a dozen 5 ft. 6 in. by 22-ft. tube mills, while a sufficiently coarse battery screen, with low discharge, might be used to double the duty of the remaining 500 stamps. High amalgamation recoveries up to 74%, as at the Princess, are obtained with coarse stamp-milling pyritic banket ore, followed by efficient classification and tube milling, and, under similar conditions, it is probable that the amalgamation recovery from Homestake ore would rise, owing to the fine reduction of pyritic particles before overflowing the hydraulic classifiers of the tube-mill circuit, and the retention of amalgam in the most efficient trap which the tube-mill circuit constitutes.

As indicated by the authors, the absence of the tube-mill circuit has militated against satisfactory work, but with the small feed given—70 tons of sand per 24 hr. to a 5 × 14-ft. tube mill—better results should be obtained by dewatering the pulp fed into the tube mill down to 28% moisture, in place of the 41.2% of moisture actually employed.

Crushing Frozen Concentrate (By N. L. Stewart).—I was called upon recently to investigate the merits of various kinds of crushing machinery for frozen ore and concentrate. The material to be crushed was frozen fine material containing hard lumps. The most promising machine seemed to be a squirrel-cage type of disintegrator. However, there was some doubt as to whether it would stand the hard blows at high speed and it was feared that the frozen material might build up on the runners. Therefore, a wagonload of frozen concentrate was put through as a test.

The test was made at the plant of the Salt Lake Pressed Brick Co., near Salt Lake City, Utah. The machine used was a 44-in. Stedman's disintegrator driven by a 75-hp., alternating-current, General Electric Co., form K motor. The motor was belted to a countershaft, from which, besides the disintegrator, a short belt conveyor was driven. The machine runners were driven at about 400 to 500 r.p.m. No clutch was used, the motor being capable of starting the machine without such a device. The machine handled the material easily and the product was as fine as the original material before freezing. When the machine was opened at the end of the test, the runners and casing were found to be clean and free from any accumulation of frozen ore. Meter readings, taken while the equipment was running empty, indicated about 26 hp., and while crushing about 44 hp. As the belt conveyor could not have taken more than 3 hp. in either case, the machine required about 23 and 41 hp. under the two conditions.

Portable Crusher and Elevator.—At many of the shallow mines in the Joplin district, the quantity of ore mined in the early life of a property is not sufficient to justify the erection of a mill, and many operators do

not care to pay the charge of $1 per ton for hauling and milling the ore at a custom plant in the vicinity of the mine. This is particularly the case with ore that is called free milling, or ore from which little middlings is produced. Such ore is well adapted to treatment in hand jigs after undergoing coarse crushing. In such instances a portable crusher, driven by a traction steam engine or a gasoline engine, is used to crush the ore. The apparatus consists of a four-wheel wagon, strongly built of structural steel and capable of withstanding the jar of the crusher. The crusher is mounted in the center of the wagon, and a wooden platform is provided at the jaw opening, upon which is piled the ore to be crushed as it comes from the mine. At the rear end of the wagon, there is a bucket elevator for raising the crushed material, which is directed into the buckets by a hopper suspended below the crusher. This elevator is driven by a sprocket and chain, driven by a belt from the crusher. The angle of inclination of the elevator can be varied by a worm-wheel on the wagon, which engages a toothed quadrant rigidly fixed to the steel frame that supports the elevator bucket line and the upper and lower pulleys. This frame is pivotally mounted, so that the entire elevator structure can be revolved until it lies parallel to the body of the wagon and above the crusher, facilitating the moving of the crusher from one mine to another.

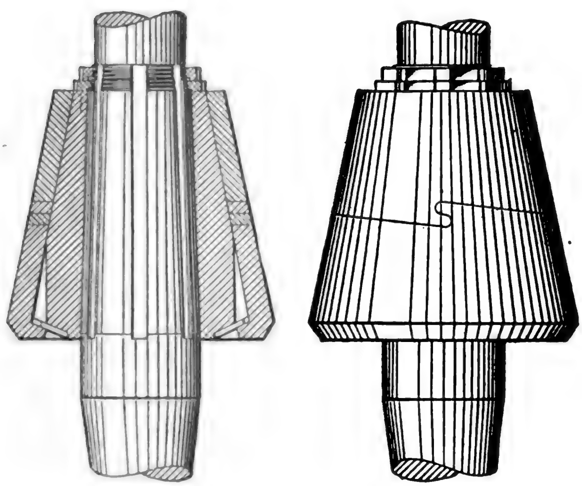


FIG. 20.—REMOVABLE HEAD FOR GYRATORY CRUSHER.

PRELIMINARY CRUSHERS

Head for Gyratory Crushers.—Rock crushers of the Gates type have a conical head by means of which the breaking is accomplished, the head having an eccentric motion which allows pieces of rock to get in

between the head and the walls of the machine, and breaks them when the eccentric motion reduces the size of the opening. Wear of this conical head is uneven and is principally on the lower half, so that the upper half may be still useful when the lower half is worn out. In order to avoid this wasteful condition, U. S. patent No. 1,066,277 has been granted to D. G. Hunter and T. G. Murton, of Germiston, Transvaal, S. A., for a conical crushing head divided into two or more parts, any of which may be taken out and replaced without the necessity of discarding the others. As shown in Fig. 20, the head is divided into parts by a spiral division, it being thus possible to put them together and lock them in such position readily and still have them available for replacement at any time. The drawing shows a two-part head.

Repairing a Gyratory Crusher.—At the Leadwood mine, of the St. Joseph Lead Co., some trouble has been experienced by the breaking of

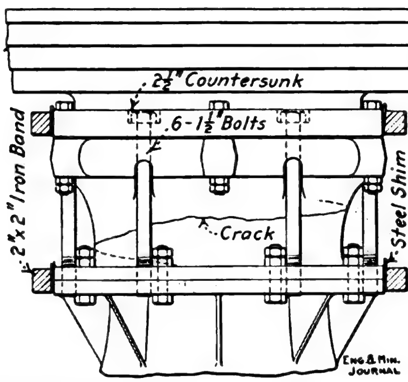


FIG. 21.—REPAIRING CRACKED SHELL OF A GYRATORY CRUSHER.

the top shell of No. 4 style B, and K Gates gyratory crushers that are used for crushing coarse ore. The shells of two No. 4K and of one No. 4B crushers have been repaired. The trouble seems to have arisen from the fact that the shell between the two rings is not reinforced by ribs. As these parts cost about $269 new, an economy was effected by repairing them at a cost of about $30 to $40 each. One of the top shells that has been fixed in the manner to be described, has been in use for four years and gives no sign of failing, while another has been in use for two years.

A crack running around the shell and almost cutting the top off the bottom part before the crack runs down into a flange, seems to be the characteristic way for a shell to fail. A typical failure is illustrated in Fig. 21. The two parts have to be bolted together, and to insure the bolts not breaking out of the flanges, bands have to be shrunk on the shell at the flanges. These hoops or bands are made by welding to-

gether bars of 2 × 2-in. iron, and six 1½-in. bolts are used to tie the top and bottom pieces together.

The reinforcing bands are made as nearly round as possible and are welded so as to make a tight fit on the shell. They are put around the flanges and any places where they do not quite touch are filled in with thin shims or wedges of steel. To shrink on a band a ring of firebrick is built around the shell, then a 1½-in. pipe with holes bored in it is run between the bricks and the shell, and is connected with the compressed-air supply pipe. Coke is piled on top of some kindlings; the fire is started; and the air turned on this improvised forge and the reinforcing bands are thus heated. While the bands are hot the wedges are driven in as tight as can be. Upon allowing the shell and bands to cool, the reinforcement shrinks tightly on the flanges so that there is no possibility of breakage by the strain from the bolts that are later put in.

Holes are bored in the flanges for 1½-in. bolts. The holes for the top flange are countersunk so as to receive the heads of these bolts, as the hopper has to be set on top of the top flange, while the holes in the bottom flange are tapped so as to catch the thread of the reinforcing bolts. Then with Stilson wrenches, the six bolts are drawn up as tightly as is possible, while held at a place between flanges to prevent turning. In this way the shell is bound tightly together and is made almost as strong as it was when new.

Electrically Driven Crushers.—In the operation of a mine one of the most important machines used is the crusher. In these days when extensive use of electricity is being made in mines it is of interest to cite defects which were experienced in connection with the operation of some electrically driven crushers, owing to the fact that it was somewhat difficult to accurately estimate the actual power that would be required. As originally installed there were two crushers each driven by a 15-hp. motor. The installation was designed to crush to about 3 in., but in order to get more ore through the mill the jaws were afterward closed to 1 in. and it was then discovered that motors were overloaded to an extent which was hardly appreciated at the time. Evidence, however, of the severity of the work was soon forthcoming in the shape of trouble from coils burning out, and if the feed was continuous, it was found that the motors were actually slowed down. This trouble persisted until, owing to the fact that the mill was being seriously handicapped because of lack of ore, it was decided to take out the overloaded motors and substitute for them motors of exactly double the power. This cured the trouble so far as the electrical portion of the equipment was concerned, but the trouble was only transferred to another part of the mechanism, for it was then discovered that too many shafts were bent on the crushers owing to the torsional strains. In order to overcome

this second difficulty the keys were taken out of the driving pulleys and shear-bolts put in their place in order to make the provision that if the crushers became choked or were fed too quickly the bolts would be bent or sheared, thus saving the shafts. By this additional safety device the crushers could be operated at a high speed and crush to 1 in. with comparatively few interruptions.

Dust-proof Housing for Dry-crushing Plants.—The most satisfactory scheme for abating the dust nuisance in mills is to house all dry-crushing machines in tight compartments. With the unseasoned wood that is available in Western mining camps, it is, however, absolutely impossible to build a plank partition, even of-tongue-and-groove material, that will remain dust-proof after seasoning and the resulting shrinking and warping takes place. This difficulty may be overcome in the following manner. All dry crushers are inclosed, ordinary boards, which need only be dressed on one side, being used for building partitions. Narrow cracks are left between the edges of the boards and strips of muslin then pasted over each joint of the partitions. The strips are not stretched tightly across the cracks but slack is left and this loose cloth is tucked into the cracks between the boards, by simply running a dull knife blade down the cracks. Then as the boards shrink with seasoning, the slack of the muslin strips is taken up without any openings developing through which dust can fly. As the loose cloth is tucked through the cracks in the partitions the appearance on the outside is not unsightly. This construction is also particularly adapted for inclosing rooms where samples are handled.

Laying Mill Dust with Water Sprays.—In the crusher house for the new West mill of the Bunker Hill & Sullivan company, at Kellogg, Idaho, the dust is almost entirely eliminated. Ordinary fruit-tree sprayers are directed over all crushers, rolls and at the discharge end of conveyor belts. The fine mist serves to collect and lay nearly all the dust. The only possible objection to the use of the sprayers is from the wetting of the floors and machinery in the plant. This is, however, not nearly so objectionable or dangerous to health as an atmosphere full of dust particles. Then, too, it is easier to clean up the moistened material than the light, dry, all-pervading dust.

ROLLS

Gripping Roll Shells on Cores.—When wet crushing is practised it is generally customary to use wooden wedges to hold roll shells tightly on the cores. The method is entirely satisfactory in such cases since the moisture keeps the wedges constantly damp and consequently tight; in dry crushing the roll shells become heated, causing wooden or iron wedges to work loose. To obviate this difficulty, the practice in the

lead district of southeastern Missouri is to shrink the shell on its core. The shell, with the two conical members of the core in position, is heated. As the shell expands, the bolts of the core are tightened as much as possible, thereby drawing the conical parts into the shell. The contraction of the shell on cooling causes it to grip the core tightly.

The equipment used at the mill of the Federal Lead Co., Flat River, Mo., for heating the shells, is shown in Fig. 22. The apparatus consists essentially of a gasoline tank *A*, made of 8-in. pipe; a firebox *B*, made of 10-in. pipe; a gasifying compartment *C*, of 2½-in. pipe filled with iron shavings, within the pipe *B*; the circular burner *E*, made of 1-in. pipe, with ⅛-in. holes, 2½-in. apart, punched in its inner circumference. Both gasoline tank and firebox are made of wrought iron; sleeves and plugs are used to make the ends tight. Charcoal is put into the firebox through a 6 × 10-in. door *G*. A 4-in. pipe, about 4 ft. long, serves as a chimney. The whole apparatus is carried in a frame, which is long enough to support both pipes. Gasoline is fed to the gasifying compart-

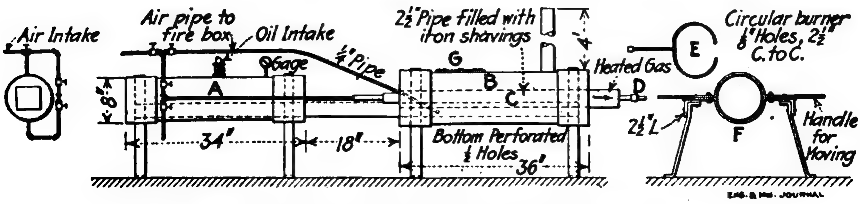


FIG. 22.—DEVICE FOR HEATING ROLL SHELL.

ment with compressed air at 80-lb. pressure. A ¼-in. pipe leading from the air main serves as a blower to keep the charcoal burning in the firebox. The burner is placed around the roll shell and the gasified mixture ignited. It takes about 10 min. to heat a shell hot enough so that it will grip the cones tightly when it is cooled. The shells give as good service as those tightened by wedges.

At the Doe Run and Leadwood mills a mixture of equal parts of coal oil and gasoline, atomized by compressed air, is fed directly to a circular burner similar to the one described above. This makes a simpler arrangement, but by the use of a gasifying tank a hotter flame is obtained and the gasoline is used more efficiently.

STAMPS

Foundations

The City Deep Battery Foundation.—In the new City Deep mill on the Rand the stamps weigh 2000 lb. each. These heavy stamps are carried upon the foundations illustrated in Figs. 23 and 24. At first glance, the

cross-section of this foundation recalls that used at the Goldfield Consolidated mill in Nevada, but this resemblance disappears upon closer inspection, for wood blocking is used below the overhanging parts of the concrete structure and the foundations are carried up as narrow pillars at each end and in the center of each 10-stamp block, which pillars, with cast-steel mountings, constitute the king posts, which, in most stamp batteries, are made of wood. These pillars are made of reinforced concrete. Upon them rests the enormous camshaft-bearing casting, and

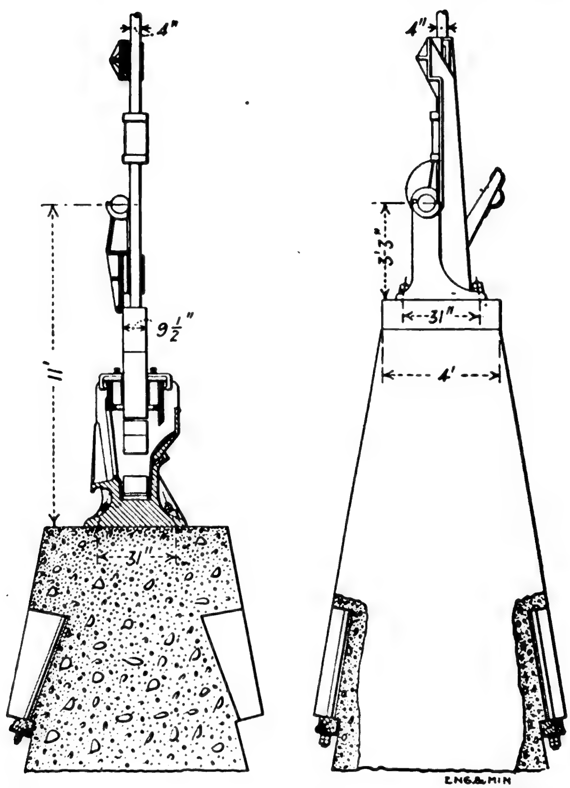


FIG. 23.—CROSS-SECTION OF BATTERY FOUNDATIONS.

the uprights which support the upper guides. Eight foundation bolts are used for each mortar casting, four in front and four in back. Besides these foundation bolts, the longer bolts that pass down through the reinforced concrete king post and hold the camshaft bearings in place, are shown in the front elevation of the battery.

In the City Deep mill, the mortar boxes rest directly upon the top of the concrete foundation, apparently, from the illustrations, which are reproduced from "A Textbook of Rand Metallurgical Practice, Vol. II.,"

without the use of a rubber pad or other packing between the bottom of the mortar and the top of the foundation. This is quite at variance with what has come to be regarded in this country as typical South African practice, for we have been given to understand that in most of the mills on the Rand, heavy anvil blocks are to be found between the mortar and the foundation. In some cases, when such a cast-iron anvil block is used,

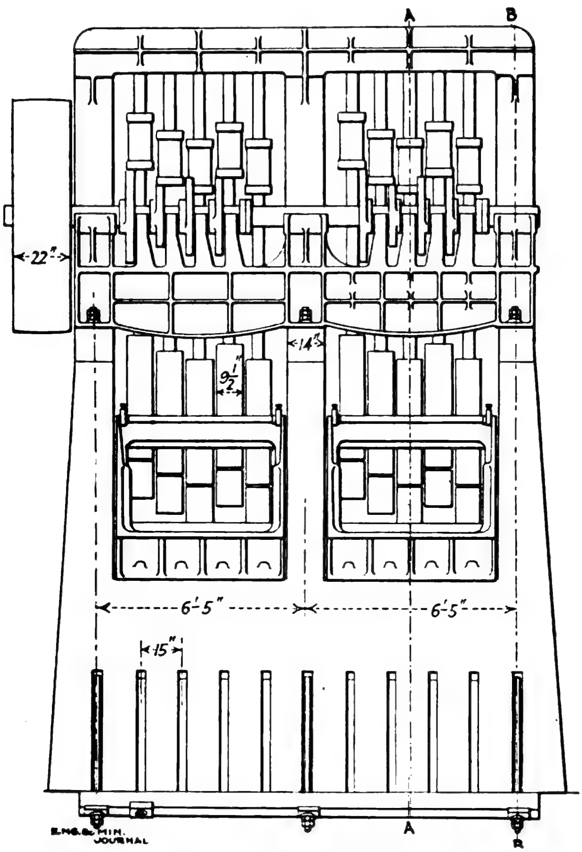


FIG. 24.—FRONT ELEVATION OF CITY DEEP BATTERY.

it does not rest directly upon the top of the foundation, but upon a series of 14 × 14-in. pieces of Oregon pine laid close together and across the battery. However, in some mills, where no anvil block is used, it is customary to place a ⅜-in. rubber cushion between the bottom of the mortar and the top of the foundation. The experience of several stamp-mill designers in this country has shown that such packing is not at all necessary; to the contrary, they regard it as preferable to place the mortar directly upon the concrete. It is stated that in South Africa,

the cost of the foundation when using anvil blocks is nearly double that when none is used.

Altering Stamp-mill Foundations.—At the Simmer & Jack Proprietary mines, Transvaal, it was found necessary to consider seriously the "re-conditioning" of the mill foundations. C. O. Schmitt, at a meeting of the Institution of Mining and Metallurgy, described the plans to permit of an increase in the weight of the stamps. In Fig. 25 are shown the battery foundations before and after the alterations. The weight of the stamps used was 1250 lb. and the average running weight, 1150 lb., while after the alterations as shown it is considered that an average running weight of 1400 lb. will be permissible, taking into account the size

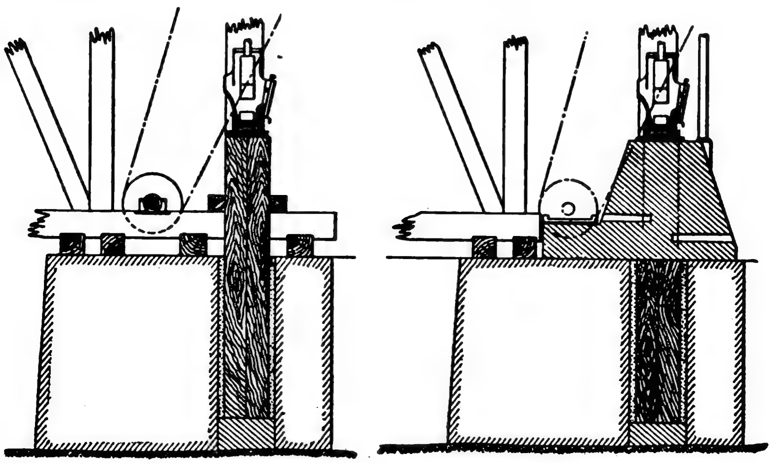


FIG. 25.—STAMP MILL BEFORE AND AFTER ALTERATIONS.

of cam shaft and stem. This represents an increase of 21.74% over the old weight and will permit of a similar increase in the stamp duty. In addition to the mere increase in stamp duty a greater efficiency is to be expected since the heavier stamp is more likely to reduce the maximum size of ore fed to the mortar box, say 2-in. cubes, in one blow to the fineness required for passing through the screen.

The diagram to the right in Fig. 25 does not show all the alterations made. Besides providing the mortar box with a substantial concrete foundation, the kingposts were cut off and are now mounted on heavy cast-iron shoes which permit of holding the posts firmly in place. The bearings of the mill line shafting are also provided with new foundations, thus making them independent of the ore bin and assuring a true line for this shafting quite irrespective of whether the bin is empty or full. The cam-shaft platform was also rebuilt and it is now carried on columns standing on the foundations, thus avoiding the vibrations always ex-

perienced if the platform is supported by the king-posts. This portion of the work, although not affecting the crushing capacity of the mill, is nevertheless important, as it enables the operators to perform their work in comparative ease and comfort.

Cost of Concrete Battery Foundation.—The literature on detail costs is scarce enough to warrant the following cost statement and description of a concrete battery foundation that was built for the Tanguay Mining & Milling Co. at Idaho Springs, Colo. The foundation supports ten 1000-lb. stamps dropping 5 in., 108 times per minute. The base of the foundation is 5 × 16 ft. The top is 2 ft. 4 in. × 12 ft. and height 8 ft. It contains 2 cu. yd. of rock and 14 cu. yd. of cement. The weight is approximately 62,000 lb. The mortars are held in place by 16 tie bolts 1½ in. × 5 ft., threaded 6 in. from the end and fitted with two nuts. The lower ends of the bolts were forged to take a 12-lb. rail. Four 13-ft. rails pass through the forged eye and connect all of the bolts, not only for the mortar but for the battery posts as well. The battery posts are held down by the bolts passing through special cast-iron angle irons. The upper 18 in. of each bolt is surrounded by 1½-in. pipe, which allows a slight springing of the bolts in case they do not line exactly with the holes in the mortar. A piece of soft rubber packing ⅛ in. thick was inserted between the mortar and the foundation. The cost as given by J. H. Haynes (*West. Chem. and Met.*) was as follows: Labor, excavating, $32.30; labor, foundation, $68.30; bolts and castings, $50; cement, $33.70; lumber, $10; gravel, $30; and rock, $7.25; or $231.55 for the complete foundation.

A Method of Securing Battery Posts.—The shoe and bracket for securing battery posts, shown in Fig. 26 and described by Francis W. Sewell (*Memoirs*, Mexican Institute of Mining & Metallurgy) was designed for the Guanajuato Consolidated Mining & Milling Co. The shoe is of cast iron and weighs 860 lb. It has a bearing surface on the concrete foundation 54 × 12 in. The shoe is secured to the foundation block by two 2½-in. rods, each 5 ft. long, threaded at both ends. The threaded portions are of enlarged cross-section, so that the diameter at the base of the thread is not less than that of the remainder of the rod. The lower ends of the rods pass through broad, square, cast-iron washers, embedded in concrete, and secured thereto by square nuts. Hexagonal nuts are used at the top end to hold down the shoe; lock nuts are shown in the accompanying sketch, but these are not necessary.

The shoe has a cavity cast in it, 24 × 10 × 6 in. deep, with rounded ends, into which the foot of the battery post fits. The bottom of this cavity is milled so as to give an even bearing surface to the post. At each end of this shoe are cast suitable projections, bored vertically with 1 $\frac{9}{16}$-in. holes to receive the ends of 1½-in. bolts from the brackets. The

two side brackets, which weigh about 100 lb. each, are recessed into the 12-in. faces of the posts, and secured in place by five ⅞-in. rods, which pass through the post from one bracket to the other. These rods are threaded at both ends. Nuts hold the brackets firmly in position. Passing vertically downward from each bracket to its respective end of of the shoe are two rods, aforementioned, with square nuts to attach them to the shoe and hexagonal nuts to attach them to the bracket. The lowest part of the bracket is 14 in. above the top of the shoe, so it

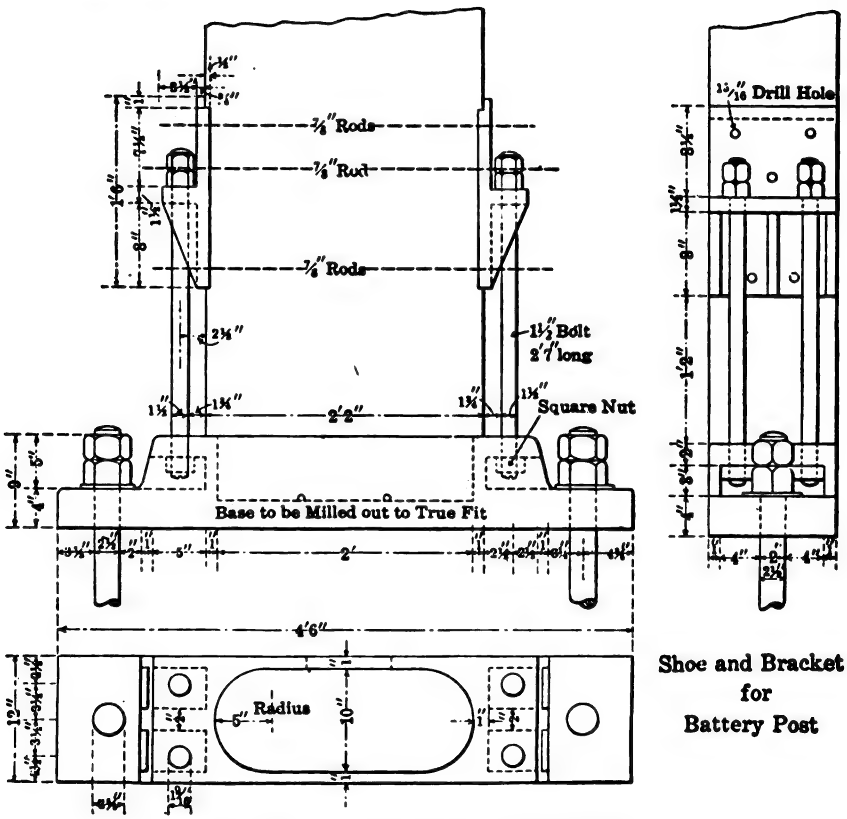


FIG. 26.—CAST-IRON SHOE AND BRACKET.

will be seen that once the post is set in position, with all the nuts tightly screwed down, there is little chance for any movement. With the exception of the lower nuts of the 2½-in. rods holding down the shoe, all the nuts are readily accessible and the rods connected with the side brackets are easily replaced.

In the erection of the battery the cavity is half filled with oil and the foot of the post is cut slightly larger than the cavity, the point being cut

so as just to enter. Some specially long rods are passed up through the brackets from the shoe, and fitted with nuts, which, on being screwed down, force the post into the cavity. This action is aided by blows with heavy hammers on the top of the post. The oil is forced into the end of the grain of the wood and passes upward some distance. This insures that no moisture can lodge in the cavity to rot the post. Frequent applications of oil are given to the posts, until it soaks in slowly, indicating saturation, after which the posts are painted.

Details of operation and design

Changes in Design of Rand Stamps.—In the Modder B mill, in the Transvaal, as the result of experience gained in the City Deep mill where crushing is being done at a cost of 1s. per ton, changes have been made in the weight and in the design of the heads and shoes. The length of heads has been reduced from 49 to 36½ in. and the diameter from 9½ to 9 in. The length of shoes has been increased from 14 to 19¼ in., the diameter remaining 9 in. These changes tend to a greater variation of crushing weight from wear. The stem remains 13 ft. long by 4 in. in diameter and the four-key tappet is 20 in. long and weighs 260 lb. against 248 lb. in the City Deep tappet, the diameter having been slightly enlarged. The total maximum falling weight has been reduced from 1978 to 1660 lb. It is apparent from this that, because of the experience with the City Deep mill, the engineers consider 1660-lb. stamps to be the most economical size to employ. With this weight there is no breakage of shoe shanks.

The 1000-lb. stamps, at the Albu mines, the West Rand Consolidated and Roodeport United mines, differ from these designs. The stems are 17 ft. long by 4 in. in diameter and weigh 721 lb. The tappets are 18 in. long and with three keys weigh 250 lb., while the heads are only 28 in. long and the shoes 16 in. long, but the diameters have been increased to 9⅞ in. and weigh 535 and 350 lb., respectively. At the City Deep mill, where Laschinger's patented cam supports are used, only two camshafts have been broken to date.

Stamp-drop Sequence (By W. H. Storms).—That the succession of drop of the stamps in the mill is of importance was recognized many years ago, and yet even at this late day there are those who entertain some curious notions about this quite simple and practical matter. If the succession of drop be not correctly arranged by placing the cams in proper position on the cam shaft, the pulp in the battery will not discharge evenly across the screen. One end of the mortar box may be empty and the stamps "pounding iron," while the other end of the mortar is so full of sand and rock fragments as to cover the dies to a depth of

several inches, and the stamps at that end not falling their proper distance, in consequence, crush a much less quantity of ore than they should and the condition inside the mortar rapidly grows worse. Or, it may be, by another arrangement of drop, that the sands may accumulate at each end of the mortar while the center dies are exposed.

It has been ascertained that the succession of drop, 1-4-2-5-3, gives as satisfactory results as any that can be devised, and yet there are those who declare in all seriousness that the drop 1-3-5-2-4 is better. Various writers on the subject of stamp milling, both in textbooks and in articles contributed to the technical press, while favoring the order 1-4-2-5-3, without reserve condemn the succession 1-3-5-2-4, or perhaps the matter is treated in exactly the reverse manner. Several writers say that mill-

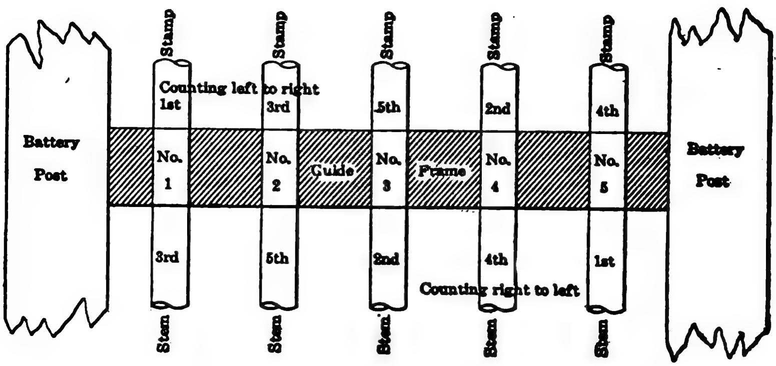


FIG. 27.—ILLUSTRATING ORDER OF STAMP DROP.

men on the Rand prefer the drop 1-3-5-2-4. As a matter of fact, these two drops, viz: 1-4-2-5-3 and 1-3-5-2-4 are identical, depending upon which side of the mortar the count commences, whether the right or the left side. Standing at the plates and looking toward the stamps the drop 1-4-2-5-3 becomes the succession 1-3-5-2-4, if viewed from the back of the mortar. In Fig. 27 this is clearly shown.

Needles for Measuring Screens (By Algernon Del Mar).—The amalgamator in a stamp mill is often at a loss to know just when to change the screens on his batteries. The apertures may wear large, but the screen may be perfectly sound in every respect. The daily samples of the tailings will go to the assayer and when a certain loss is noted, the millman, in looking for the cause, may find that the screen is letting through particles of a size that carry with them too much valuable metal. Ordinary sewing needles are fairly uniform in size and the sizes from No. 1 to No. 9 can be bought in any dry-goods store. Knowing the sizes of these needles the millman may be able to anticipate the assayer and thus save money for the company. For example, it may be found

by experiment that when the apertures of a screen have worn to, say, 0.0390 in., the screen must be changed. A No. 3 needle is about this size and when this needle will pass freely through the screen, it is time for a change to a new screen. Table X shows the size of the head of different needles, as measured on the large side of the head by a micrometer, and the equivalent or corresponding screen.

TABLE X.—SCREEN APERTURES AND EQUIVALENT NEEDLES

Needle number	Largest diameter of head, in.	Equivalent screen
1	0.0735	10 mesh.
2	0.0428	16 mesh; No. 3 slot.
3	0.0395	16 to 18 mesh; No. 3 to No. 4 slot.
4	0.0355	20 to 24 mesh; No. 4 to No. 5 slot (nearer the last).
5	0.0335	20 to 24 mesh; No. 4 to No. 5 slot (nearer the last).
6	0.0300	24 mesh; No. 5 to No. 6 slot.
7	0.0265	24 to 26 mesh; No. 6 to No. 7 slot.
8	0.0230	26 mesh; No. 8 slot.
9	0.0203	29 mesh; No. 9 slot.

Wear of Screens in Single Stamp Mills (By Algernon Del Mar).—Those who have had charge of single-stamp mills know how excessive is the wear on the screens. The cost of new screens often becomes a serious factor. The times lost in changing screens, about five minutes for each, must also be taken into account. In the square-mortar type the back screen is, I believe, unnecessary, for it does little work and the screening capacity of the three other screens is greatly in excess of the crushing capacity. It is hardly necessary in the round type of mortar to extend the screen completely around the mortar for the same reason, and because of the fact that the greater the screen surface the greater the cost for repairs in screens for other reasons than the wear, due to the impact of ore particles, as will be explained.

A screen that may last 20 or more days on a five-stamp mortar with a given discharge, may only last two or three days on a single-stamp mortar with the same discharge. The causes of this are many; it is due to the proximity of the die to the screen, in other words to the width of the mortar at the discharge level; to the angle at which the ore is propelled from under the shoe to the plane of the screen frame; to the height of discharge, which determines the aforementioned angle, and, that which my experience points as the principal cause, the bending of the screen outward and inward, due to the suction caused by the rising and falling of the stamp. This action cracks the screen even when there is little other wear. For this reason I have found that the cheaper, blued-steel,

needle-punched screen is the most economical, as the bending does not appear to crack this class of screen any sooner than the wire or punched iron, and as they wear out in three or four days, there is no reason to use a more expensive screen for the same tonnage. My experience points to a better output with these thin screens made of blued steel.

M. P. Boss proposed a narrow circular mortar, having the die set below the screen opening as a remedy. As this will cause the discharge to be proportionately high, the purpose of this single stamp, that of giving a high tonnage, is neglected. Mr. Boss' mortar is designed to take the wear of the flying particles on the inside of the mortar, perhaps by a false lining, but he is reducing the capacity of a proportionately high discharge, and if this discharge is brought high enough, the crushing capacity of 5 one-stamp batteries will no more than equal a five-stamp battery, in which case the advantages all lie with the latter type of mortar.

The choice between the single and the five-stamp unit, where the amalgamation of the gold is unimportant, lies with the single-stamp mill, for it is the amount of ore crushed which alone is considered. This being the case, a narrow mortar with a low discharge is required, the economical depth being a subject for experiment. The increased capacity from a low discharge may more than offset the extra cost of screens and the time lost in changing them. If the amalgamation of the gold is the important consideration, then the advantage is all in favor of the five-stamp unit. There is no reason why the die should be below the mortar opening, for the same effect can be produced by a chuck block covered with sheet iron on the inside, to take the wear. This gives the advantage of lowering the discharge at will by simply changing the chuck, while with a deep mortar it may be necessary to take the die out and insert a false bottom under the die to bring it to the desired height.

I am inclined to believe that the screen, if inclined more away from the mortar, would help to make the angle of contact of ore particles with the screen of less importance and consequently lessen the wear on the screens, and if the upper portion of the mortar above the screen discharge was enlarged, a freer expansion and contraction of air, due to the suction of the stamps would result, which would prevent the pulsating movements in the screen.

A Scheme for Reducing the Mesh of a Screen.—In remote mining districts the mill superintendent frequently has occasion to use a screen of smaller mesh than he has on hand, and to secure this screen from a factory may require several weeks. A method used by the Vermont Copper Co. for reducing the mesh of a screen is simple and gives fairly satisfactory results. The screens upon which it was used were of the perforated type, with elliptical holes 1×2 in. A wire about $\frac{1}{8}$ in. in diameter was threaded through the openings, diagonally across the screen,

thus reducing the openings to one-half the original size. The wire can be fastened easily and securely at the edge of the screen either by notching the screen, or running the wire through the wood frames, if such are used.

Battery-screen Frame.—A battery-screen frame in use at several of the mills of the Mother Lode district, California, is shown in Fig. 28. The advantage of this arrangement is in the simplicity of the method of changing screens. No tacks have to be pulled or old rusted screen torn loose. To remove the outer frame *B* which holds the screens in position, it is only necessary to loosen the wooden button at *A* and slip it from the grooves in which it is fitted. The entire operation of changing screens need, with this arrangement, require only a couple of minutes. To hold the screen more firmly a narrow partition is sometimes put in the frame from *C* to *C′*.

The screens used in the batteries of the North Star mills, Grass

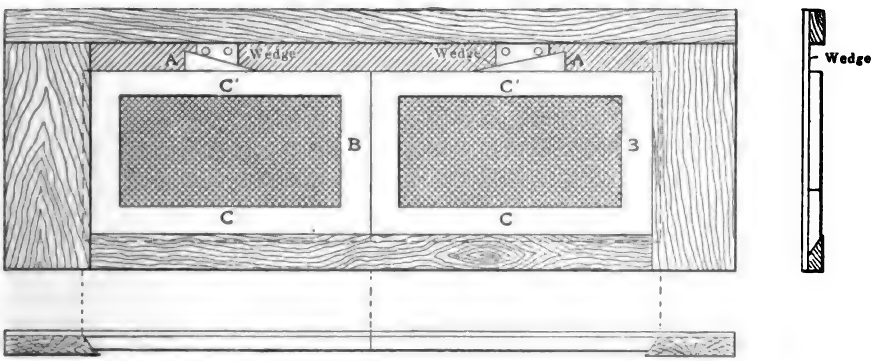


FIG. 28.—BATTERY-SCREEN FRAME.

Valley, Calif., are punched with round holes within squares of 1 in. on a side. Between punched areas unpunched bands of metal, $\frac{1}{8}$ in. wide are left, as reinforcing. A tear, starting within the punched area, does not usually extend across the band. These screens have a life of 22 days as compared with 10 days for the usual type. The screen frame is made of wood 2 in. square, the inner rectangle measuring 4 ft. 9 in. by the height of the screen. To the back of the frame strips of 2 × $\frac{1}{8}$-in. strap iron are screwed so as to extend $\frac{3}{4}$ in. into the inner rectangular space. The screen is held against the inner side of this strap-iron rim by a rectangular frame made of $\frac{3}{4}$-in. angle iron, the outside dimensions of which permit its fitting within the wooden frame. The angle-iron frame is held in place by six buttons.

An Improved Chuck Block.—Where inside amalgamation is done by plates on the chuck blocks, the difficulty caused by the accumulation

of sulphides or other heavy particles on the tops of the chuck blocks may be done away with by the simple expedient of making the upper angle of the block a sharp one, thus insuring a steep slant on all parts of the plate. Daniel McKinley, amalgamator of the Laurentian mine at Gold Rock, Ont., designed such a set of blocks, making 3 sizes, 4, 5 and 6 in, respectively, to keep pace with the wear of the dies.

False Mortar Bottom for Increasing Height of Discharge.—In certain stamp mills on the Rand, instead of using chuck blocks of different heights to compensate for the wear in the dies false bottoms are used in the mortars for raising the dies. These false bottoms, described by G. O. Smart, in "A Textbook of Rand Metallurgical Practice," consist of a steel plate about 3 in. deep, which is placed under the die in the bottom of the mortar-box cavity. As the dies become still more worn, another false bottom may be used, or coarse gravel may be placed under the false bottom, and thus both false bottom and dies may be raised. The use of a false bottom necessitates the removal of all the dies, while if the chuck blocks of different types are used, it is not necessary to delay crushing for so long a time, when compensating for the increase in discharge height.

Stamp Heads.—There is no question as to the advisability of having stamp heads made of steel instead of cast iron. A few years' use will wear down the lower edge of a cast-iron head to the breaking point. The removal of broken stem stubs is usually a difficult and tedious job, where cast-iron heads make the use of a pinch of giant powder inadvisable. Particularly is the job difficult if a stem must be removed without removing the stamp head from the mortar. The difference between the cost of cast iron and cast or chrome steel is a mere $25 in a five-stamp mill. This difference is soon offset by saving of time and useless efforts.

Lawton's Chilled Iron Stamp Shoe.—The lower edges of ordinary steam-stamp shoes quickly wear away, resulting in the formation of a convex face. This wearing continues until the lower portion of the shoes becomes quite round and therefore inefficient. Instead of crushing the ore, the shoe rams it or pushes it away, resulting in a great decrease in the output of the mill. But when the shoe is so formed that the outer edge of the wearing face is harder than the middle portion, the sharp particles of quartz or other rock will wear away the middle of the face as fast as the outer edge. The action of the stamp is more effective when the bottom face of the shoe is slightly concave. Shoes of cast iron, formed by pouring the metal into a mold having a chilling member, have been found most effective, and to have the longest life. They virtually consist of a central core and an outer shell, the shell being much harder than the core. A patent (U. S. No. 1,012,664) has been granted to Charles

L. Lawton of Hancock, Mich., covering shoes of this type and methods of casting them. According to the claims of the inventor the output of ore per stamp when equipped with a shoe having a hard outer edge and softer middle portion, and the life of the shoe, are both much greater than those of a shoe of uniform hardness equal to or even greater than the hardness of the outer edge of the shoe of the former design. This shoe may be used until worn away, while the usual shoe of uniform hardness throughout often becomes practically useless when less than one-fourth of its volume has been lost.

Troublesome Battery Shoes (By Lyon Smith).—Frequently in a large stamp mill a shoe becomes detached from a boss head and for a few minutes is not detected by the man on shift. The shoe may lodge in the mortar box in such a manner that the boss head drops on the shank of the shoe, tending to form a projecting ring on the inside of the opening into which the shoe fits. This is especially true when the shank of the shoe comes in contact with the inside periphery of the boss head, and is more likely to happen if the shoe is new and correspondingly heavy; for an old and lighter one would in most cases soon become dislodged from such a position. Often it is impossible to drive the shoe on again as the projecting ring of metal tends to cut the wedges to pieces. If the shoe is picked up in an attempt to drive it on, it is generally but a few hours before it becomes dislodged again. Shooting the boss head from the stem and substituting a new one may be resorted to, but the usual practice is to chip out the boss head. This is an awkward job and necessitates hanging up the battery from two to three hours. A good way to obviate this prolonged hangup is to open the mortar box, remove the detached shoe and substitute an old and lighter shoe, decreasing the height of drop of this particular stem to about 4 in., slightly less than the length of the shank of the shoe. Wooden wedges should not be used in driving on this shoe as a metal-to-metal contact is desired. The mortar box should then be closed and the five stems dropped as usual for a few hours, taking care to feed closely, when it should be hung up and the old shoe driven off. By this procedure the projecting ring of metal will be almost or entirely removed, and the new shoe can then be driven on in the usual way.

Removing Broken Stems from Stamp Heads.—Stamp shoes have to be removed so frequently from the stamp heads that they do not remain in place long enough ordinarily to become so firmly wedged into the head as to cause great difficulty in their removal. But on the other hand the stems are seldom removed from stamp heads except when the stems break. Then it is usually the case that a small end of the stem is left in the stamp head, where it has become so firmly attached that considerable force is necessary to dislodge it. This is often accomplished by firing a small

charge of dynamite in the head. This practice has the disadvantage that stamp heads are often cracked from the blasting.

An improved type of stamp head is described by G. O. Smart in "A

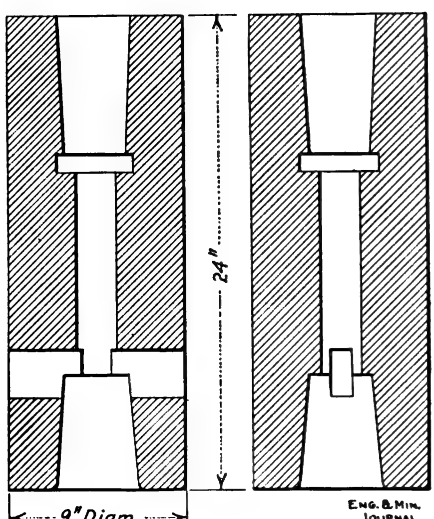


FIG. 29.—STAMP HEADS WITH BORED CENTER.

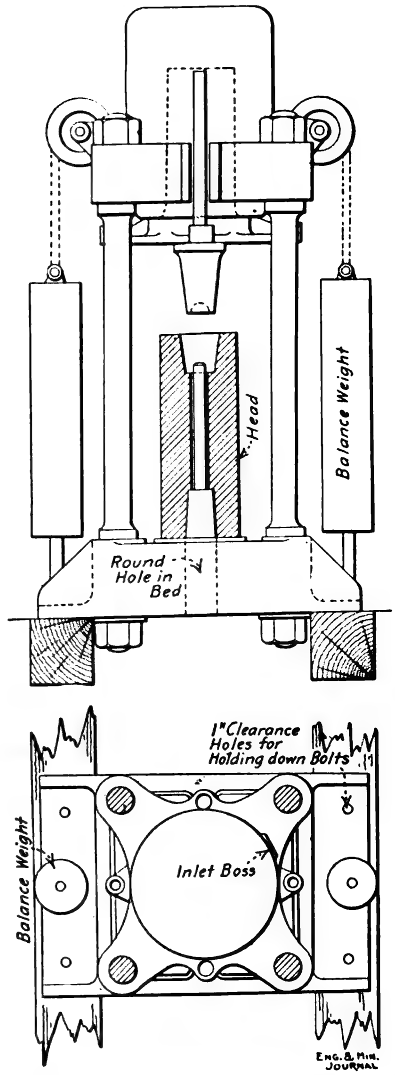


FIG. 30.—HYDRAULIC PRESS FOR REMOVING BROKEN STEMS FROM STAMP HEADS.

Textbook of Rand Metallurgical Practice," and is used in some of the large stamp mills on the Rand. This stamp head is made with a conical opening at each end for the reception of the stem. There is a smaller

cylindrical opening connecting these two openings, so that there is a continuous passage from one end of the head to the other, as shown in Fig. 29. This makes it possible to remove a broken stem in a hydraulic press, Fig. 30, especially equipped for the purpose.

When a stem is broken off and the short end cannot be readily removed from the stamp head, the stamp head is set upon the bed plate of the hydraulic press so that the broken end of the stem is directly over a hole provided in the bed plate. That end of the head in which is the broken stem is placed downward. A rod, nearly as large as the bore used in the stamp head, is dropped down upon the piece of stem. This rod extends up into the larger conical opening in the upper end of the stamp head. The plunger of the hydraulic press is tapered so as to fit easily within this conical opening. The plunger is brought forcibly down upon the rod in the stamp head, and as the pressure is increased, this rod forces the broken stem out of the bottom conical opening in the stamp head so that it drops through the round hole in the bed plate of the press.

Compensating Weights for Stamps.—Stamp shoes gradually wear away, and on account of breakages, the stems become short, and also the heads or bosses are worn by abrasion of the pulp in the water box, all of

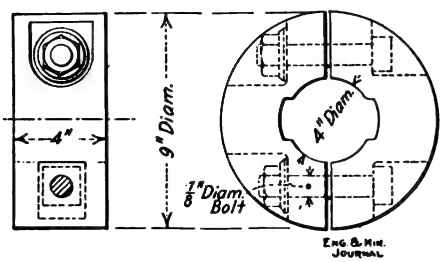


FIG. 31.—STAMP WEIGHTS USED ON THE RAND.

which cause the stamps to lose weight. Compensating weights are often used to make up for the decrease in weight in order that the full crushing capacity may be maintained. The early forms of these compensating weights were crude in design, and often a head was placed upon the top of the stem above the upper guide or an extra tappet was used above the one by which the stamp was lifted. A simple type of compensating weight is described by G. O. Smart, in "A Textbook of Rand Metallurgical Practice," and illustrated in Fig. 31. These compensating weights consist of a split disk. which weighs about 60 lb. and is of the same diameter as the tappet. These weights are clamped with a bolt on either side of the stem, and placed as low down as possible. If a compensating weight be placed above the tappet, it is useful in taking the place of the tappet-set formerly employed when altering the height of stamp

drop. Since the use of compensating weights in a 200-stamp mill will increase the crushing capacity by 10 to 20 stamps, it is obvious that the small amount of additional labor and trouble required in using them is well repaid by the advantages gained.

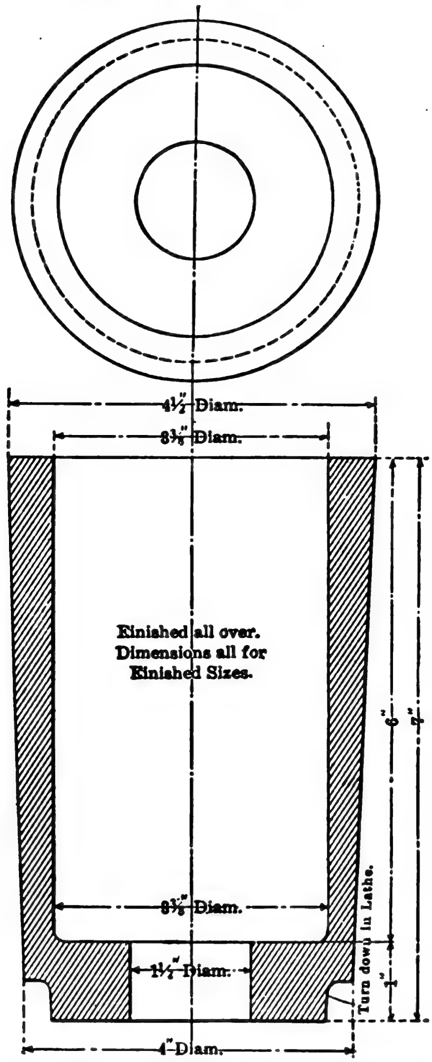


FIG. 32.—A BUSHING FOR STAMP STEMS.

A Bushing for Stamp Stems.—The stems of the stamps in the mills of the North Star Mines Co., at Grass Valley, Calif., are not attached to the boss heads in the usual manner. The stems are without taper at either end, and are held in the boss by a tapered bushing. The bushing

is made of steel, the details of design being shown in Fig. 32. The outside of the bushing that fits into the boss is tapered, while the inside cylindrical space for the reception of the stem is of the same diameter throughout. The bushing is made in two pieces, each lacking a sixteenth of an inch of being an entire semi-cylinder, and is grooved for a width of ⅛ in. by a milling saw, from the top almost to the base, in order that the bushing may have a certain amount of spring. The bushing is placed on the stem, then wrapped with a piece of canvas and dropped into the boss with the stem reaching as near to the bottom of the bushing as possible.

The stems usually break just above the boss head, and when this happens a new boss and shoe are set without taking the stem from the guides. The broken piece of stem is driven out by a drift key, or in case of that means failing, is shot out with dynamite. A shoulder is turned in the base of the bushing so any burring of the metal by the drift key will not project far enough to touch the side of the boss, and so impede the bushing being driven out. About 20% of the bushings are broken in driving the broken pieces of stem out of the boss, but this cost is largely offset by the promptness with which a new boss and shoe can be fitted to a broken stem.

Myers' Stamp-stem Guide.—A guide for stamp stems which has met with favor among millmen is the design known as Ideal. This

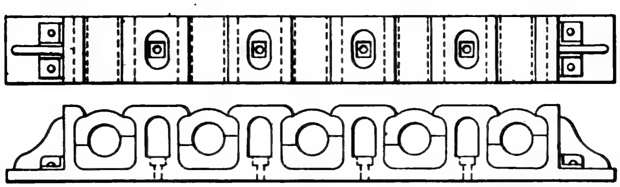


FIG. 33.—A GUIDE FOR STAMP STEMS.

consists of a cast-iron frame holding five split journals, the outside of which is formed in wedge shape, substantially as shown in Fig. 33. The advantages are that no bolts or screws are used in the journals and they can be put in or taken out by hand whenever required. The wedge shape allows wear of the stem in the journal to be taken up automatically. The device is made by G. W. Myers, of San Francisco.

Moyle's Finger for Gravity Stamps.—The ordinary method of hanging up stamps, with a camstick, has been generally considered a rather annoying one and often a little dangerous. Broken arms and crushed fingers have not infrequently resulted from its careless use. In this connection it is interesting to note the device shown in Fig. 34 by means of which the stamp can be hung up instantly by one standing on the lower floor in front of the mortar. The principle and operation are

clearly shown in the drawing and are readily understood. The faces *A* and *B* are finished with a piece of leather or rubber packing, to prevent crumbling of the iron through vibration. At the point *C*, a wooden block is placed for the same purpose. The advantages are obvious. It was invented by E. H. Moyle, of Los Angeles, Calif., and patented under U. S. No. 1,040,235.

Behr's Stamp-lifting Device.—Hans Charles Behr, of Johannesburg, South Africa, has patented (U. S. No. 1,012,187) a means of lifting stamps

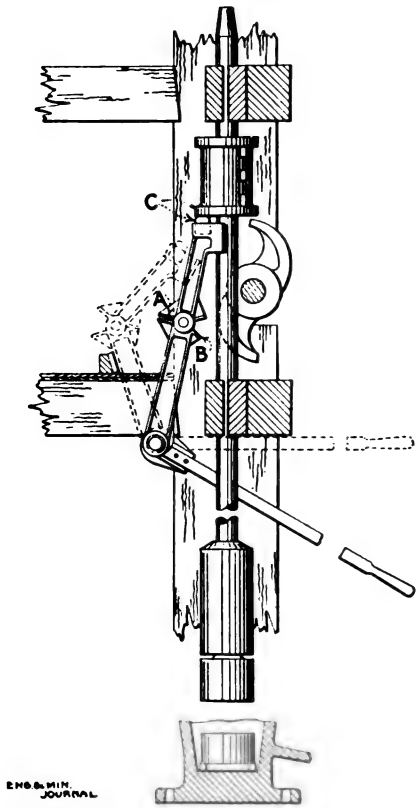


FIG. 34.—FINGER FOR GRAVITY STAMPS.

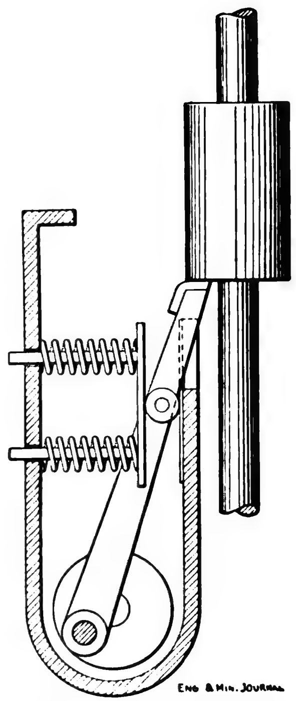


FIG. 35.—BEHR STAMP LIFTER.

in which the lifting itself is done by a dog driven by a shaft, instead of by the usual method of a cam mounted directly on the shaft. Suitable means of release are provided by which the dog is withdrawn to allow the stamp to fall. There are several mechanical variations of the idea, Fig. 35 illustrating one.

Lifting Gravity Stamps.—In Fig. 36 two devices are shown for attaching a chain-block hook to a stamp stem when it is desired to raise the stamp

for any purpose, such as changing a die. The operation of the device shown in Fig. 1 is evident from the drawing. When the ring in the device shown in Fig. 2 is pulled upward, the rectangular piece of metal turns upon the bolt, biting into the stem with sufficient force to enable the stamp to be raised without slipping.

Cam-shaft Collar (By J. H. Oates).—The drawing of a cam-shaft collar which does not loosen on the shaft is shown in Fig. 37. It was designed by Charles Harbottle, of Guanajuato, Mex., and has proved to be far superior to the collars that have to be shrunk on the shaft. As

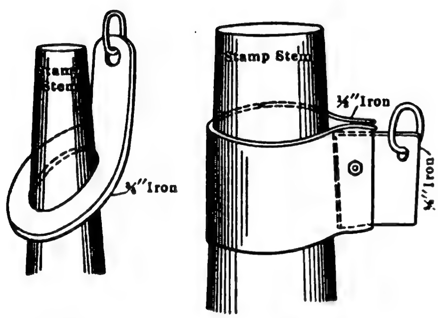


Fig. 1 Fig. 2

FIG. 36.—SELF GRIPPING CLAMPS.

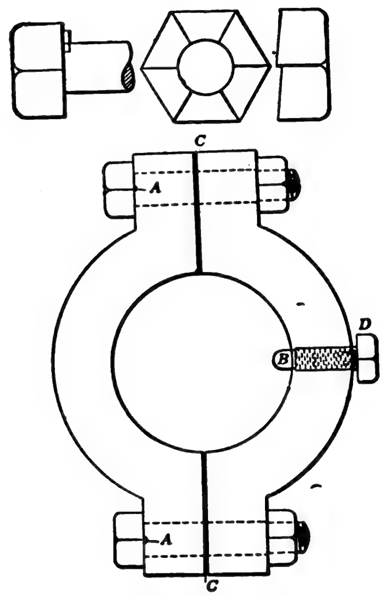


FIG. 37.—NONSLIPPING COLLAR FOR CAM SHAFT.

shown, it is a split collar made of 2 × 2-in. iron, turned out to the exact size of the shaft. In machining the collar a shim $\frac{1}{8}$ in. thick is bolted between the two halves at *C;* this gives $\frac{1}{8}$ in. to be taken up when the collar is in place. The bolt shown at *D* is $2\frac{3}{4}$ in. long, with $\frac{3}{4}$ in. at the lower end turned as shown to $\frac{5}{8}$-in. diameter. This end enters a hole that must be drilled in the shaft with a ratchet. The nuts are made of case-hardened tool steel. The teeth shown must be cut before hardening. Bolts have the ordinary 1-in. hexagonal head. A small hole is drilled through each bolt just under the head, as shown at *A*. A pin inserted through this hole and a corresponding groove cut in the collar keeps the bolt from turning. To take the collar off it is best to cut the heads from the bolts as it requires some time to turn the nuts backward. The

side of the collar that comes in contact with the cam-shaft bearing is faced.

Application of Power

Position of Driving Power for Stamp Mills (By Algernon Del Mar). —Battery posts of reinforced concrete such as described in *Eng. and Min. Journ.* Sept. 25, 1909, are the only type which will allow of a vertical pull on the cam shaft, therefore it needs no bracing and may be independent of the ore bin or other foundation. Fig. 38 is a composite drawing of

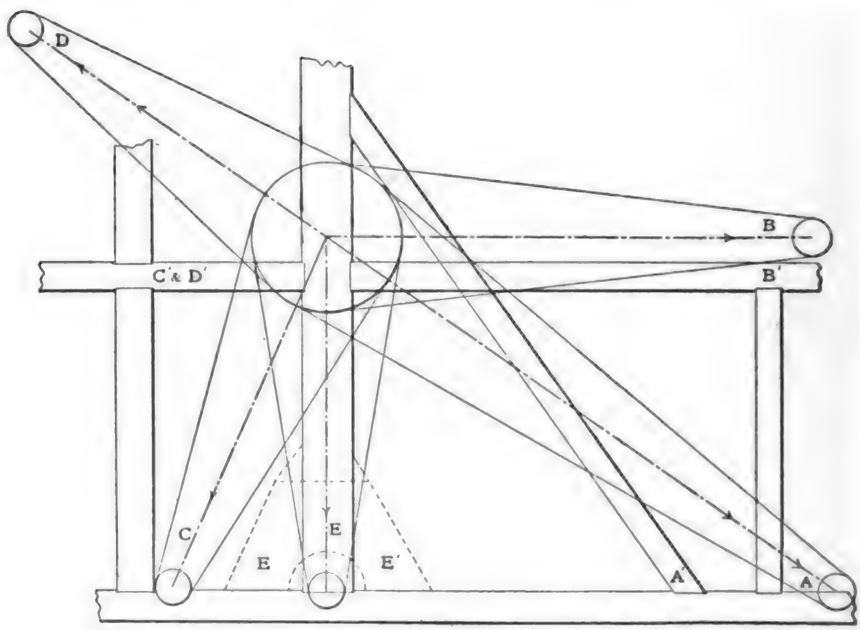


FIG. 38.—STAMP-MILL FRAMES AND METHOD OF DRIVE.

the different forms of construction. The A-frame is omitted. *A* is the direction of pull for an angle-brace construction and *A′* the brace; *B B′* shows the front knee; *C C′* the usual form of back knee; *D D′* the overhead form where the slack belt is uppermost, and *E E′* the concrete battery post. A 3-ft. semi-circular casting forms a hollow through the battery posts and mortar blocks in which the driving-shaft boxes are placed. Since the publication of the idea of concrete battery posts and cam-shaft cushions I have discussed the idea with many practical millwrights who without exception have declared it practical, and some were enthusiastic in its praise.

Geared Motor for Stamp-mill Drive.—A motor for driving stamp mills has been recently placed in use. It has several features which make it

valuable for the hard work and unfavorable conditions which attend stamp-mill service. The motor is mounted in a cradle, which in turn is mounted on a heavy base-plate. The cradle carries a back shaft, to which the motor is geared. The back-shaft pulley runs at slow speed, so that it can be belted directly to the bull wheel of the stamp mill, thus eliminating the jack-shaft and saving space, belting, the loss of power due to belt slippage, and the expense of the jackshaft. The pulley is outside the bearing on the cradle, so that the belt can be easily removed.

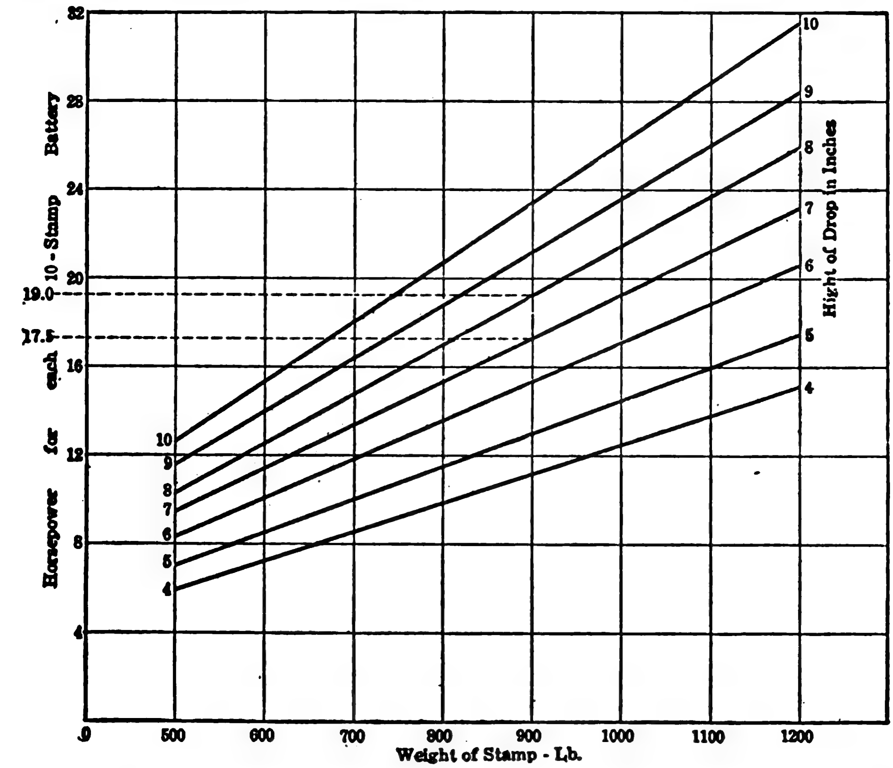


FIG. 39.—HORSEPOWER DIAGRAMS FOR STAMP BATTERIES (INCLUDES ALL FRICTION).

The motor, gear and pinion can be removed without disturbing the line-up of the pulleys or handling the belt. The back-shaft bearings are split and can be inspected or renewed after merely slackening the belt. The gears are inclosed in a dust-proof case and run immersed in oil. This installation can be supplied for driving mills of from 3 to 20 stamps, with stamps weighing from 800 to 1250 lb. One motor is ordinarily used for each battery and is shut down when the battery is not in use, thus avoiding a waste of power. It is a product of the Westinghouse Electric & Manufacturing Co.'s shops.

Power Required for Stamp Batteries.—The power necessary to drive stamp mills in units of 10 stamps is readily calculated from the diagram, Fig. 39, prepared from the notes of E. H. W. Westwood, of Melbourne (*Aust. Min. and Eng. Rev.*, Oct. 5, 1900). The chart is figured on a basis of 90 drops per minute. The following example explains the method of using the diagram: Required: The horsepower necessary to drive a 10-head battery of 900-lb. stamps, 90 drops per min., height of drop being 7 in. Take the figure 900 under the "weight of stamps," follow this line vertically until it intersects the 7-in. drop line, then proceed horizontally to the left of the diagram, where it will be found that the required power for each 10-stamp battery is 17.5 hp. Another use of the diagram is in finding the change in horsepower, by varying the height of drop without changing the weight of stamp; for example, with a 900-lb. stamp, in changing the drop from 7 to 8 in., the increase in horsepower for each 10-stamp battery is from 17.5 to 19 horsepower.

Power Required for a Stamp Mill (By H. S. Knowlton).—In connection with a test of the power consumption at a gold mill in the Idaho Springs district, the Central Colorado Power Co., of Denver, recently made a determination of the effect of shutting down individual stamps upon the performance of the engine driving the installation. The equipment of the mill comprised the following machinery:

Crushing and Sampling Plant.—One 9 × 15-in. Blake crusher; two Vezin samplers, 36 × 24 in.; two 10 × 6-in. bucket elevators, 60 ft. and 53 ft. 7 in.; two Mitchell sample crushers and one sample grinder. Fine-crushing department: Twenty 800–850-lb. gravity stamps, 7-in. drop, 88 drops per minute.

TABLE XI.—POWER REQUIRED FOR A STAMP MILL

	Average indicated horsepower from engine cards					Average indicated horsepower from engine cards			
	Without crusher or sampler		With crusher and sampler			Without crusher or sampler		With crusher and sampler	
Number of stamps dropping	Without generator	With generator	Without generator	With generator	Number of stamps dropping	Without generator	With generator	Without generator	With generator
20	46.3	49.3	52.3	55.3	11	35.4	38.4	41.2	44.2
19	45.0	48.0	51.0	54.0	10	34.2	37.2	39.9	42.9
18	43.8	46.8	49.8	52.8	9	33.0	36.0	38.7	41.7
17	42.6	45.6	48.5	51.5	8	31.8	34.8	37.5	40.5
16	41.4	44.4	47.3	50.3	7	30.6	33.6	36.3	39.3
15	40.2	43.2	46.1	49.1	6	29.4	32.4	35.0	38.0
14	39.0	42.0	44.6	47.9	5	28.2	31.2	33.8	36.8
13	37.8	40.8	43.6	46.6	4	27.0	30.0	32.6	35.6
12	36.6	39.6	42.4	45.4	0	22.2	25.2	27.6	......

Concentrating and Regrinding Department—Seven concentrating tables; one vanner; three tailing pumps, one mill-water pump; one 5-ft. Colorado Iron Works grinding pan and one agitator, not operated during test. Miscellaneous: One 3-kw. lighting generator, 115 volts; one 4-kw. generator, 6 volts, for energizing mercury traps and agitator and one 17-kw. dynamo not operated. The mill is operated continuously, except on Sundays. The power requirements with varying number of stamps are shown in Table XI.

Holding Down the Cam Shaft on a Stamp Battery (By Claude T. Rice).—It has been found practically useless to try to hold down the cam shaft of a stamp battery by means of caps on the boxes. As a consequence, it is the general practice to use the boxes without caps. This allows much vibration and the babbitt of the bearings wears away quickly under the continual hammering, and has to be renewed frequently. To avoid

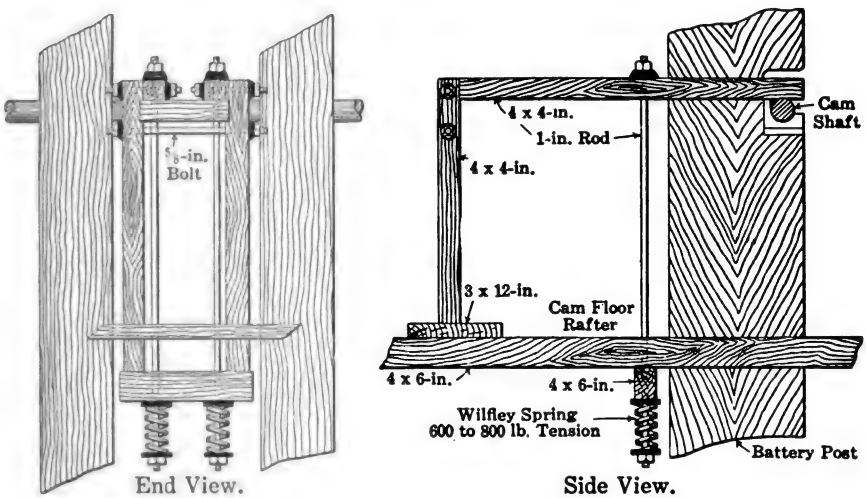


FIG. 40.—CAM-SHAFT DAMPING DEVICE AT MONTANA-TONOPAH MILL.

the delay and expense of rebabbitting so often, soft-iron cam-shaft bearings, which are not babbitted, are sometimes used; the bearing is replaced when it is worn out. The following method of damping the vibration of the cam shaft in its bearings is used at the Montana-Tonopah mill with great success, and it is said to have proved equally effective at the Goldfield Consolidated mill. As shown in Fig. 40, a 4 × 4-in. piece of wood is pressed down on the top of the cam shaft by means of a spring from a Wilfley table, as this is the spring that can most easily be obtained in a mining camp. It is strong enough to give the desired tension, and of good enough quality to resist the strain of the constant vibration. It is placed on a 1-in. rod and tightened so that there is a pressure of from

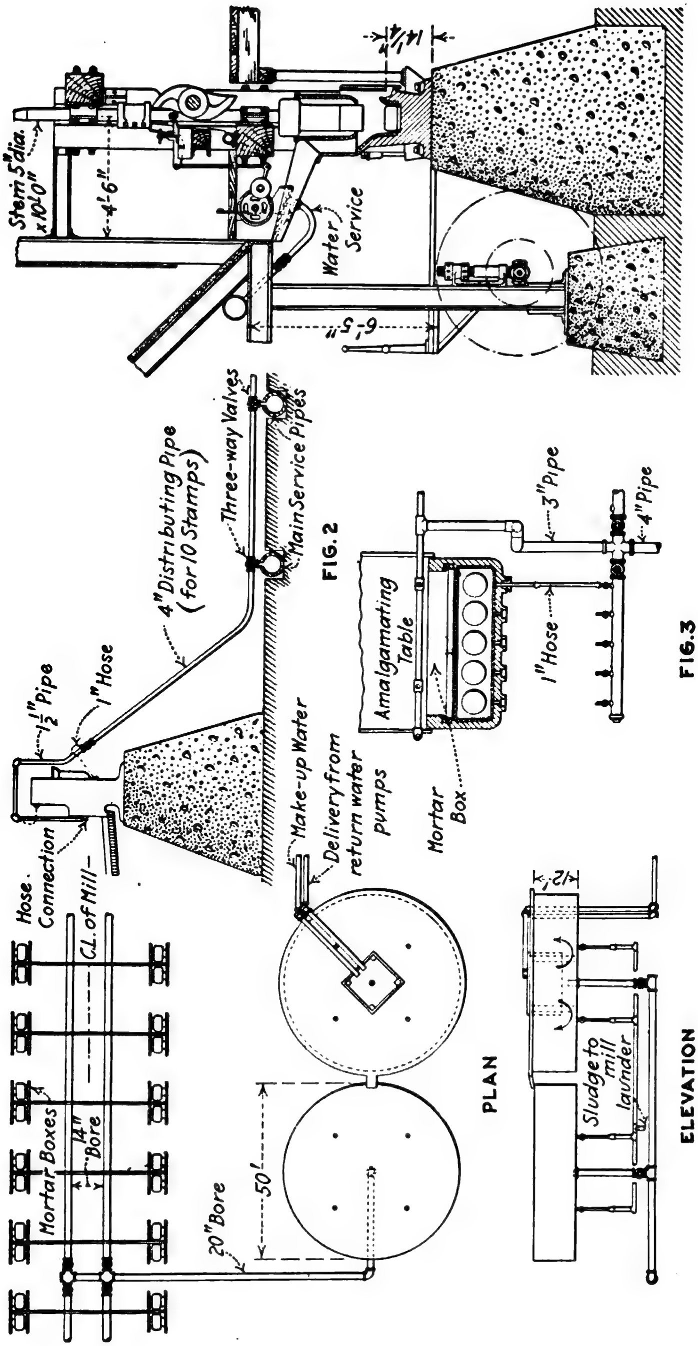


FIG. 41.—RAND METHODS OF SUPPLYING STAMPS WITH WATER.

600 to 800 lb. exerted downward on the top of the cam shaft. These damping devices are placed over each bearing of the cam shaft and damp the vibration of the cam shaft effectively. At the Montana-Tonopah mill, before this device was installed, the cam-shaft bearings had to be rebabbitted every six to nine weeks, as the vibration rapidly chipped out the babbitt metal. Now the shaft has been running for four months with the holding-down spring device, without a single bearing having to be rebabbitted. The same result is effected at the Tonopah-Belmont by means of a railroad car spring footed against a shoulder cut in the battery post, and pressing an oak block 12 in. long against the cam shaft. This method of damping the vibration is also found quite effective, and seems to be somewhat simpler, although railroad car springs would have to be specially ordered at most mines.

Water Supply

Stamp-battery Water Supply.—The quantity of water required in a stamp mill depends upon the method of working, and as a general rule, it may be taken, that the amount of water used in the large gold mills of the Rand equals, according to C. O. Schmitt, in "A Textbook of Rand Metallurgical Practice, Vol. II," 7 tons of water per ton of ore milled but it may vary from 4 to 10 tons.

The water supply should be under a constant head or pressure, so that extra attention for regulating the quantity due to variations in the head will not be needed. As it is, the water supply requires continual adjustment to follow the changes in the screen used. In the common practice on the Rand the greater part of the mill water is returned from the cyanide plant, and this is highly alkaline, while the make-up water, that is, the quantity added to compensate for the loss in the cyanide plant, and which may be as much as 20% of the total, contains mostly acid. Where the two are mixed a reaction takes place, resulting in a precipitate. Arrangements should be made to perfect settling the precipitate prior to entering the mill service pipes, as the resulting incrustation will ultimately choke the pipes. This can be effected by providing storage vats with a chamber where the two water supplies are mixed before entering the body of the vat. The mixing should preferably be in the center of the vat, so that the water can leave by a peripheral launder, gravitating to the second vat, from which it should be drawn at a point removed as far as possible from the inlet, so as to give every possible facility for settling the precipitate and other impurities present. Referring to Fig. 41, Fig. 1 shows the arrangement of a plant designed on this principle. The capacity of the storage vat should be equivalent to at least the 24-hr. supply, so that in case of failure of the return-water pumping plant the mill need not be stopped.

The main service pipes from the storage vat to and in the mill should be provided in duplicate, each capable of handling the whole supply, so that when one pipe is being cleaned out the work may proceed. The distributing pipes leading from the main service pipes to the mortar boxes should be so arranged that they can be taken out and replaced by a spare set in the shortest possible time, thus affording an opportunity for thoroughly cleaning the pipes taken out. All pipes should be of ample size to allow of a certain amount of incrustation before requiring cleaning, but, on the other hand, they should not be too large, as the velocity of the water will then be very low, and the solid matter carried in suspension will settle more readily, increasing the rate at which incrustation takes place. It is a case of finding the happy mean between conflicting considerations. A velocity of 2½ to 3 ft. per sec. when the pipes are clean would seem reasonable. Fig. 2 shows an arrangement for distributing pipes that will give satisfactory results. Fig. 3 shows a detail of the piping at the mortar box.

Up to within recent years the water feed to the Rand mortar box, used to be by one or two pipes, either over the top or through the front of the box. The method now generally used consists of a separate jet for each stamp, so arranged that the water tends to strike the face of the die slightly behind the center line. It is assumed that the jet of water so applied will remove all very fine material from the face of the die and aid in washing it through the screen, thus increasing the capacity of the stamp. The nozzles used should be at least 1-in. bore, as they are liable to choke, due to incrustation. Where there are two points at which water is fed to the mortar box, the main supply being through the back of the box by means of an individual jet to the face of each die, and an auxiliary supply to the front of the box above the head, the five jets are connected to a header by small closed pipes, the regulation being effected in the pipe feeding the header by two valves or cocks. One valve is generaly used for cutting off the supply, while the second valve, or cock, is used for regulating purposes, thus avoiding the necessity of again regulating the supply every time it is cut off. The same method of regulation is used for the auxiliary supply through the front of the mortar box. In Fig. 4 is shown the method of supplying water to a Nissen stamp battery.

Feeders For Stamp Batteries

A Simplified Challenge Ore Feeder (By Henry B. Kaeding).—From the situation of any feeding device, it must necessarily be exposed so that there is every opportunity for grit to find its way into the train of gearing by which the feeder plate is driven. As a result, the wear on the

parts is severe and if the driving mechanism is at all intricate, the operation of the feeder is easily disarranged. The simpler the driving mechanism the less annoyance is caused by wear. The general construction of a simplified Challenge ore feeder is shown in Fig. 42. The improvement consists in removing all pawls, springs, blocks, etc., in fact, the entire central feed mechanism including the spider, leaving only the empty disk wheel. A piece of $\frac{5}{8} \times 2\frac{1}{2}$-in. Norway iron is then forged into two pieces, the details of which are shown, and are drilled and bolted as indicated. The piece when assembled should be a loose fit on the disk-wheel rim. The spring for returning the driving lever must be strong enough to overcome any jamming of the lever on the disk wheel. Any blacksmith can assemble this working part in an hour and it can be

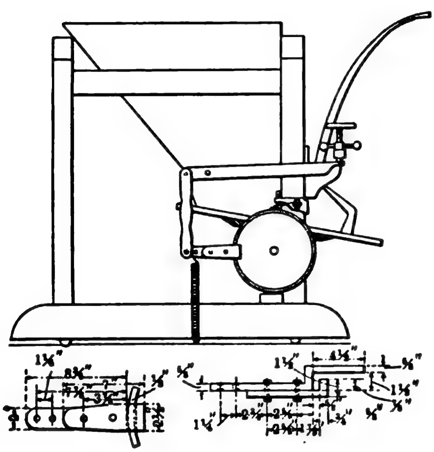


FIG. 42.—ROTATING MECHANISM FOR A CHALLENGE FEEDER.

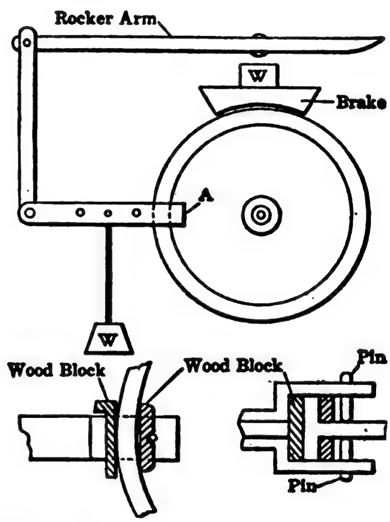


FIG. 43. —AUTOMATIC FEEDER AT LIBERTY BELL MILL

attached to the feeder in 10 min. This device is positive in its action, never requires adjusting, wears a year or more, costs practically nothing to replace, and can be taken off and replaced in 15 min. I do not know who should have credit for designing this feed; it is known to many who have seen it at El Oro and elsewhere.

(By Charles A. Chase).—The feeder mechanism described in the preceding article and also stated to be used at Bodie, Calif., was devised and patented in 1899 by Charles D. Hooper, foreman of the Liberty Bell mill at Telluride, Colo. A bad feeder mechanism caused extraordinary losses of time and he found in this design complete relief. It has continued in use to the present time with the utmost satisfaction, and I

doubt if its equal is available at present. Mr. Hooper sold the rights to use this design to the Edward P. Allis Co. which, I believe, made little use of it, perhaps not appreciating its full value. The original construction was slightly different from that shown in Fig. 42. The additions consisted of two shoes of wood or iron which fitted against the inner surface of the ring and a flat piece of wood which fitted outside the ring. A brake, consisting of a weight or a shoe, rested on the top of the wheel. I am not sure whether this has been found a necessity or not. Instead of the spring we have commonly used a weight on the arm. I mention this simply as an interesting fact in connection with the device. Fig. 43 illustrates these details.

The Hamil Ore Feeder.—At the Searchlight mill, in Nevada, an ore feeder is in use that was lately designed and patented by Charles Hamil, the millman. It is said that the operation of the feeder has been satisfactory. The feeder consists of a hopper about 18 in. wide and 3 ft. long, the bottom of which is a broad belt, equal in width to the bottom opening of the hopper. This belt is mounted on rollers and is moved forward by a wheel which is driven by a trip actuated by one of the falling stamps in much the same manner as the familiar drive of the Challenge feeder, but it is claimed that there is more latitude in the adjustment of the belt- than there is in the disk-discharge feeder. The ore comes down from a chute to the hopper, thence passes to the belt, which carries it forward, discharging into the feed opening of the battery. The new feeder is being tried out side by side with the old-type feeders at the mill of the Searchlight Mining & Milling Company.

Battery Ore Feeder, Rio Plata Mill (By Alvin R. Kenner).—A simple efficient feeder in use at the Rio Plata mill, Guazapares, Mex., is illustrated in Fig. 44. The first feeder of this design was put in to replace a Challenge-type machine undergoing repair. The installation was rather crudely put together, being intended simply as a makeshift, but worked so well that the remaining batteries were similarly equipped. The feeder has given much better service with but a fraction of the attention and repair costs formerly required. The construction of the feeder is readily apparent from the illustration. A collar or ring on the stamp stem governs the feed as the amount in the battery increases or decreases, on the same principle as a Challenge-type feeder. This collar may be held in place in several ways, but an arrangement similar to that used in fastening a tappet will be found most serviceable.

It should be noted that the feeder is hung out of center, *i.e.*, the center of the pan is several inches back of a vertical line passing through the point of support on the cam floor. When the collar *A* on the stamp stem

strikes the inclined iron in front, *B* the pan is thrust sharply back, but immediately returns to its former position owing to the manner in which it is hung. The links at *E* are made large enough to allow the necessary movement, which is not more than 2 in. A simpler construction in the rear support may be used if small hand wheels are not available. A bolt may be run through the timber and lock nuts placed on the other side. This arrangement has been used and gave no trouble, but requires more time should a change in the character of the ore require readjustment of the position of the pan. The front piece *B* is separate from the triangular shaped iron *C* to facilitate renewal. When worn a new piece is simply bolted on in place of the old one. In the two and one-half years which these feeders have been in use at Rio Plata these are

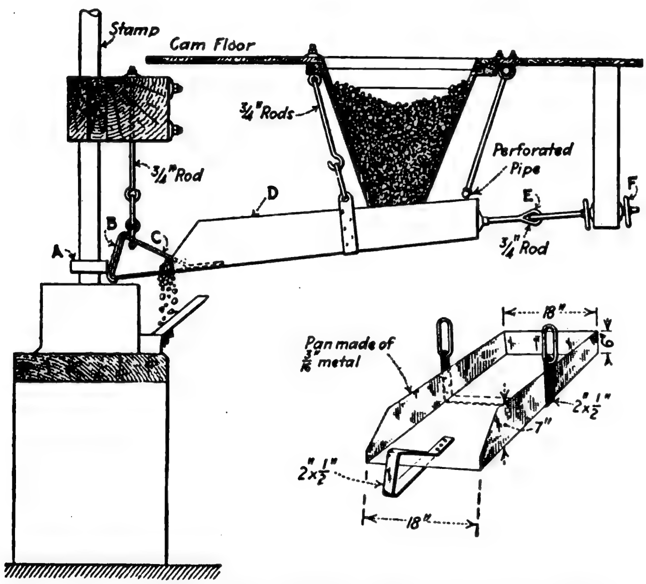


FIG. 44.—BATTERY ORE FEEDER USED AT RIO PLATA MILL.

the only parts which have been renewed. This style of feeder will outwear two of the Challenge type, and the regularity of feed is equally satisfactory. Besides the small first cost of the feeder, the small amount of attention and repairs required, and the wide range of adjustment offered by the manner in which it is supported, it has an added advantage which will be found most convenient whenever necessary to make repairs in the back of the battery. The front and middle supports can be unhooked and the pan lowered out of the way in a couple of minutes, thus leaving plenty of room to work.

A back piece and water pipe are shown in the sketch but are not used

at the Rio Plata as the ore is dry; the feeder shown in the sketch was designed for a wet-ore feed. The perforated water pipe keeps the bottom of the pan wet, which is necessary to obtain a proper feed. The feeder may not do as good work on wet as on dry ore, but C. E. Ballow, a former master mechanic at the Rio Plata, who invented the feeder and installed those in use, states that he has tried them on wet ore and that they work equally well. There are no restrictions on the use of this feeder and it should prove a welcome change where trouble is experienced with feeders of complicated mechanism.

Reducing Wear on a Feeder Drive (By Algernon Del Mar).—The millman at the Ford Bidwell Consolidated mill, Fort Bidwell, Calif.,

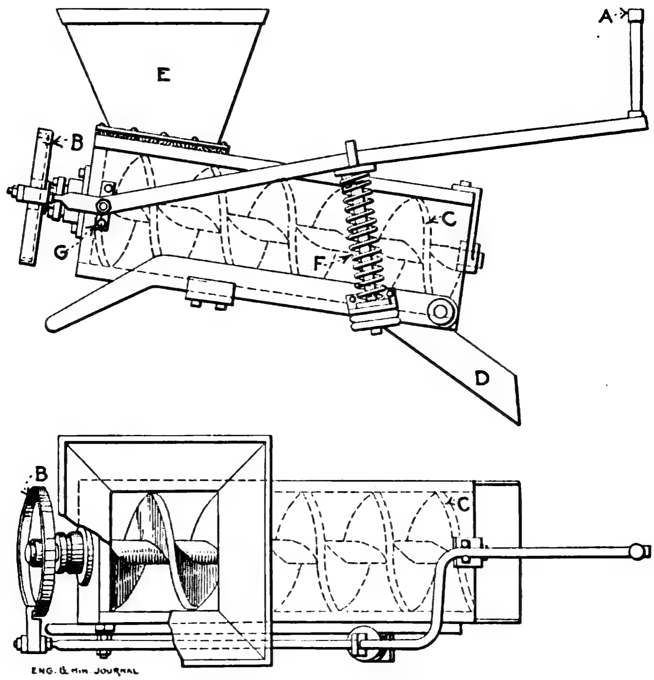


FIG. 45.—STAMP-MILL FEEDER.

has introduced a useful but perhaps not a novel device for a cushion on the feeder bar of the automatic feeder. The end of this bar is usually horseshoe shaped, to which a strip of leather is riveted. The leather often comes loose and is then a great annoyance, for the iron of the bar and the collar on the feed stem wear rapidly. The device is a ring of leather the pieces of which are riveted together. This fits loosely around the stem and has caused no trouble at all. Owing to its freedom to move around the stem it tends to wear evenly.

Sutton's Feeder for Stamp Mills.—A feeder for stamp mills, which operates on a principle somewhat different from the usual one, has been designed by Thomas Sutton, of California, and patented under U. S. No. 1,047,589. Its operation is apparent after consideration of Fig. 45. The movement is imparted at *A* by the tappet and communicated by the lever, pivoted at *G* to the wheel *B*, which is grasped by a plain wrench grip. This allows free movement in one direction but carries the wheel with it in the other. Fixed to the shaft of the wheel is a spiral screw *C*, which moves with it, taking the ore from the hopper *E* and delivering it to the spout *D*, which leads into the mortar. A spring *F* returns the lever to the normal position after having been depressed by the tappet. The device is simple, has few parts to get out of order, and appears entirely practical.

Behr Battery Feeder.—H. C. Behr, of Johannesburg, has patented (U. S. No. 1,047,396) an automatic feeder for stamp batteries. As shown by Fig. 46, the ore from the bin is supplied to the battery by both a

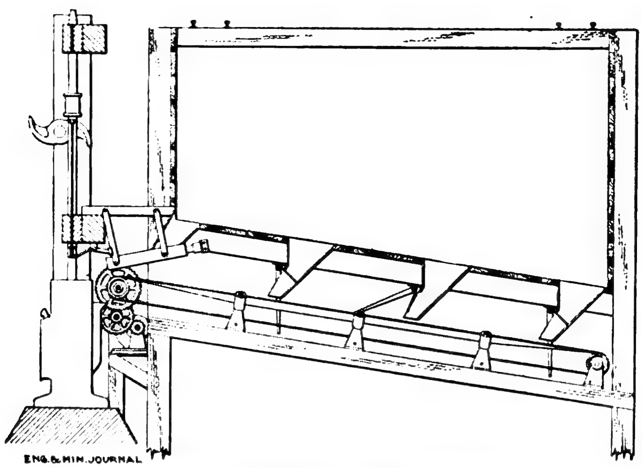


FIG. 46.—BEHR BATTERY FEEDER.

conveyor belt and a shaking feeder of the bumper type. The conveyor belt, as the primary feeder, maintains a constant supply of ore to the battery at slightly under its normal capacity. Any deficit is made up by the shaking feeder which is actuated by the impact of the tappet of one of the falling stamps upon a rod. The length of this rod is so regulated that the tappet will not reach it except when the drop of the stamp is longer than normal, owing to an under supply of ore in the battery; when the deficiency has been made good the drop of the stamp is shortened sufficiently so that the auxiliary feeder will cease to work.

COMMINUTING MACHINES

Grinders and Roller Mills

Grinding-pan Practice at Great Fingall.—From investigations of the grinding-pan practice at the Great Fingall mine, Western Australia, E. Jensen (*Monthly Journal*, Chamber of Mines of Western Australia, January, 1913), states that better results were noted when corrugated shoes and dies were used because of the greater grinding surface obtained thereby. Another important factor was the pan speed, better grinding results, with a lower consumption of power per pan, being obtained with a speed of 54 r.p.m. than when the pans were running at 58 to 60 r.p.m. Further, with extremely hard shoes and dies, the grinding results were not at their best. A general diminution of the grinding efficiency was noted as the shoes and dies wore. This was not the case with shoes and dies of ordinary hardness, the grinding efficiencies with them being approximately even during their whole life. By increasing the thickness of shoes and dies from 3½ in. to 4½ in., the life was lengthened at a comparatively small increase in the cost of each set.

Huntington Mills and Their Operation (By Claude T. Rice).—The great trouble with Huntington mills is their high cost of upkeep, and the close watching they usually require to keep them in good condition. The cost of upkeep and repairs on Huntington mills is directly dependent upon the condition in which they are maintained. If mullers are allowed to become rough and flat, as the die rings are uneven, then pounding is sure to result, and the mill racks itself to pieces. Most of these troubles appear to be due to the fact that the mills are improperly fed. The cause of the wrong feeding can be traced directly to the prevalent practice of feeding a thin pulp to the mills instead of a dewatered product and adding the required amounts of water below the mills. Most Huntington mills are treating an oversize product, coming from rolls, that has first been sent to jigs or coarse tables to remove as much of the free mineral as possible before it is ground finer. As a result the consistency as well as the amount of the pulp varies and it is impossible to keep a regular load on the Huntingtons without giving them an undue amount of attention. But if the product from the concentrators is dewatered, as it would be, were it being sent to rolls, and if this dewatered product is stored in front of the Huntington mills in a bin fitted with automatic feeders, say of the plunger type, all trouble from over- or under-feeding is done away with, as a regular and constant feed can thus be maintained. This is the practice at the Silver King mill, at Park City, Utah, where it is probable that Huntingtons are run at less cost of upkeep than at any other mill in the country.

Even if the Huntington mills are fed regularly, the dies and mullers

will wear unevenly, but fortunately the mullers get flat mainly on account of the die rings wearing unevenly. Consequently the problem is to keep the die rings true. Shoes are sent out with Huntington mills as they come from the factory, but they are seldom used to true up the die ring, as they greatly decrease the capacity of the mill. These shoes are cast-iron segments of a circle having the inner diameter of the die ring, bolted to a yoke similar to the one that carries the muller. This shoe, as it goes around in the mill, tends to wear down the high places in the die ring. Unfortunately it grinds slowly, and if there is much quartz in the pulp, the die wears unevenly from top to bottom, quartz accumulates in the places where the shoe is countersunk to receive the bolts, and wears the die faster than the metal of the shoe. The shoe itself, therefore, requires frequent truing up, or new irregularities will appear while the older ones are being removed. The capacity of the mill is decreased

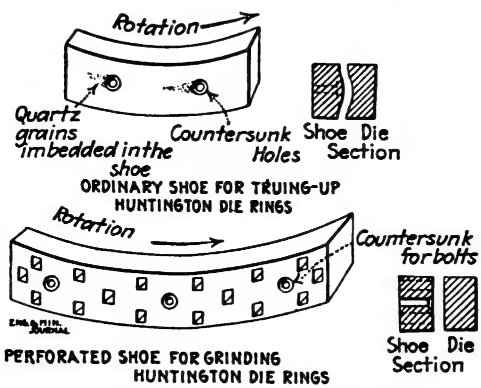


FIG. 47.—HUNTINGTON MILL SHOES.

as the muller does not come entirely in contact with the die throughout the whole of its face.

This drawback in the ordinary shoe has been overcome by J. W. Thompson, superintendent at the Silver King mill, by having soft cast-iron shoes (see Fig. 47) made with holes scattered regularly over the surface for the quartz grains to accumulate in. The quartz grains then train back from the holes and become imbedded in the soft cast iron of the face, so that soon the whole surface is made up of a number of sharp quartz grains imbedded in the cast iron. As the grains are evenly scattered, both laterally and vertically, over the surface of the shoe, the wear on the ring is even, and owing to the fact that the quartz grains cut faster than does the ordinary cast-iron shoe, the shoe does not have to be in the mill as long a period as the old-style ones. In order to concentrate the wear as much as possible on the high places in the die ring, the shoe is made longer than in the case of the ordinary shoe,

and on that account, too, the wear is faster than if an ordinary shoe were used. Such shoes are kept ready on a yoke, and as soon as any tendency toward chattering is noticed in the mill a muller is taken off and a shoe and yoke put on in its place. As the die rings are kept smooth, there is no reason for the mullers to pound and they wear evenly. As the mullers do not pound, bolts to not work loose, and breakages from that cause do not result. Therefore, the mills in the main need merely inspection, and no large repair crew is required to keep them in operation.

Mechanical Feeders in Bunker Hill & Sullivan Mill. (By John Tyssowski).—Types of feeders mechanically operated from eccentric

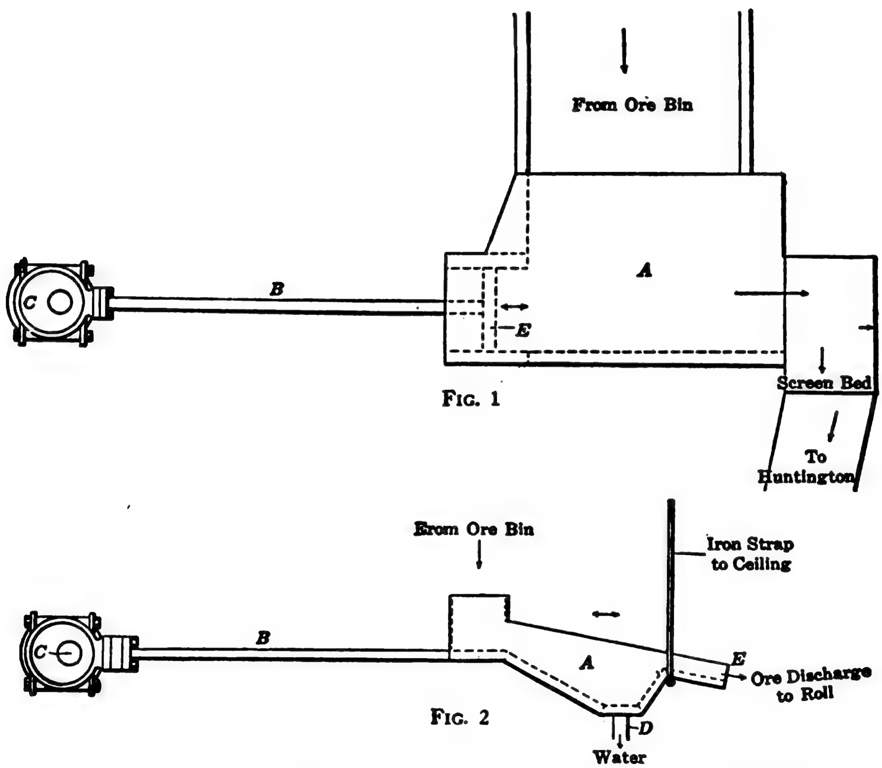


FIG. 48.—MECHANICAL FEEDERS USED IN BUNKER HILL AND SULLIVAN MILL.

shafts, that are used in the new concentrating mill of the Bunker Hill & Sullivan company, at Kellogg, Idaho, are shown in Fig. 48. Fig 1 is a feeder for the Huntington mills. There is an incline from the crushed-ore bin through which the ore slides into the chamber *A*. At one end of this chamber there is a piston *E*, connected by the rod *B* to the eccentric shaft *C*. Motion of the shaft causes the piston to slide back and forth

in the chamber, thus forcing the ore into the incline to the Huntingtons. There is a screen bed over the incline, so that only the finer material is fed to the Huntingtons, the coarser material passing on as indicated in the drawing. The device shown in Fig. 2 is used for feeding rolls. The ore passes by gravity into the hopper *A*, which is connected by the rod *B* to the eccentric shaft *C*. Motion of the eccentric shaft causes the hopper to shake back and forth; the hopper is hung by iron straps fastened to the ceiling. Water drains off from the hopper through the pipe at *D* and ore is discharged to the rolls through the spout *E*. The hopper is built of sheet steel and is wood lined to prevent wear. Besides giving a constant feed, this device also acts as an efficient dewaterer. It is run at 200 r.p.m., and is given a 2-in. shake. Both arrangements are automatic in their action and insure a constant feed to the grinding machines, which is a decided advantage. The rate of feed can be easily regulated by controlling the rate at which the eccentric shaft is revolved.

The Tube Mill

Rand Tube Mill Practice.—The most effective work in all crushing of hard grains is done by impact, and the falling pebbles in a tube mill are in this respect like the falling stamp in a stamp mill, says W. R. Dowling in "A Textbook of Rand Metallurgical Practice, Vol. I." The greater the number and the greater the height of drops the more crushing is done. Similarly the greater the diameter of the tube mill the greater will be the height of drop, and the higher the speed within certain limits the more drops per minute. The style of liner is of importance in determining the speed necessary to obtain the maximum height of drop. Should the liner be smooth, as is the case when steel or iron plates are used, then the mass of pebbles tends to slip, especially with a light load, and the revolutions per minute must be increased as compared with the more irregular flint liners. The rapid wear of cast-iron or steel liners appears to be due to the slipping load of pebbles, mixed with sharp-edged sand grains, which grinds away the liner. The life of tube-mill liners on the Rand appears to be far shorter than is the case in other mining districts, such as Kalgoorlie, and the fact may be attributed to the hard abrasive nature of banket ore and to the high pressure at which tube mills are worked in these fields. It is of utmost importance that tonnage and moisture of feed be maintained at correct standard. With a large dry feed of sand the mill appears to become overloaded at the feed end. The ideal pulp feed appears to be of that consistency which permits of the sand particles adhering to both pebbles and liner, so that whenever a pebble strikes other pebbles or the liner there are grains of sand to receive the impact and be crushed. A series of tests by G. O.

Smart led to the conclusion that, for Rand conditions with a 22 × 5½-ft. tube mill, 400 tons of sand containing 39% moisture by weight should be fed per 24 hr. With a small feed, 200 tons of solids per day, 27% moisture in the pulp was found to give the best crushing. The reason for this is believed to be that with a large tonnage of exceedingly thick pulp the mill cannot discharge at an adequate rate and becomes overfull, so that large masses of sand and pebbles are projected at once instead of the pebbles individually, as would be the case with a more dilute pulp feed. With a small thick feed, however, overcrowding in the mill does not take place, and more efficient crushing is obtained than with a small feed of solids as a diluted pulp.

Dry Tube Milling (By H. T. Durant).—In some milling operations it is necessary to crush the ore while dry, in all the various stages up to,

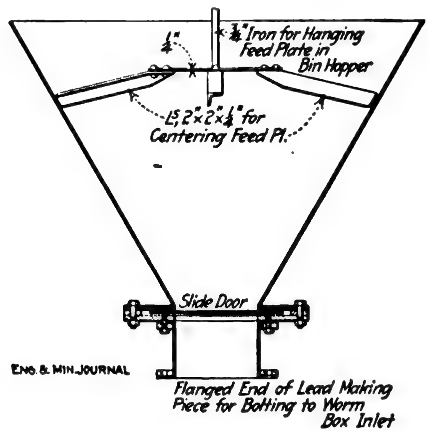


FIG. 49.—FEED HOPPER FOR DRY MILLING.

and including the final tube milling. In such cases unless the moisture in the tube-mill feed be carefully watched the output of the mill will be seriously affected, or the mill may become more or less clogged. In a particular case the ore came from the calcining furnaces and was allowed to cool before being sent to the tube-mill bin; this roasted ore was of a hygroscopic nature, and experience showed that if the required fineness of the product were to be maintained the output of the tube mill had to be halved when the moisture in the feed rose to about 1%. At one time the moisture in the mill feed was unfortunately allowed to rise to more than 1½%. After about 6 hr. running, the interior of the mill began to "make up" or to become lagged with ore in which the pebbles were firmly embedded, the whole mass adhering to the mill liners; this naturally reduced the working diameter of the mill and put most of the pebbles out of action. This lagging up of the in-

terior of the mill is, of course, gradually evident from the change in the sounds during running. The clearing of the mill is not easy, especially when it has only one manhole door. The mill slowly cleans itself if fed with sharp dry sand.

Dry tube milling need not be dusty work if the feed and discharge ends are well arranged. Where the feed end consists of a horizontal worm box feeding through the hollow trunion, as shown in Fig. 49 the flanged intake on the upper side of the worm box can be set a few inches vertically below the discharge end of a hopper-bottom bin of any convenient size, such as one to hold a 24-hr. run. At the bottom of the hopper bin will be the ordinary slide gate for regulating the feed, and between this and the intake of the worm box will be, flanged and attached by setscrews, a short making-up piece made of about 10-lb. sheet lead; this will be more convenient than a stuffing box.

By this method of construction it will be found that, since the bin itself may settle or rise ever so little with its varying loads, the line of the worm box will not be affected, as the lead making-up piece connecting the bin to the worm box will respond more readily than a stuffing box could. A 1-in. hole should be made in each of two opposite sides of the lead making-up piece. These can be closed with corks or other plugs. When adjusting the feed by the sliding gate which is just above this lead making-up piece, it is an advantage to be able actually to see the feed, and this is easily done by removing both corks and placing an electric light at one hole, then looking through the other.

A steel feed plate, centrally hung in the hopper portion of the bin at about 2 ft. above the slide gate and centered by four short pieces of angle iron riveted to it, will insure a steady feed from the bin to the worm box without any chance of the ore in the feed bin blocking or bridging. If the cross-section of the hopper at the level of the feed plate is 3 ft. square, then the plate itself will be about 14 in. square. It is advisable to guard the entry to the tube mill feed bin by screens, so that under no circumstances can anything enter it which is too large to be easily passed along by the feed worm.

Tube-mill Power (By H. E. West).—Considerable misconception exists with regard to the power necessary to drive tube mills. Usually the power is underestimated. It is therefore of interest briefly to examine the recorded power consumption of any installation, such as that given in Table XII which is the August, 1910, record of the El Oro Mining Co., El Oro, Mexico.

In the third column is given the rating of the motor driving the mills, each of which has its own motor, except Nos. 3, 4 and 5, which are driven from a line shaft. The last three have bevel-gear drive in contradistinction to the plain gear of the others. In size the first three are approxi-

mately equal to the Krupp No. 3 mill (old classification), 4½ × 16 ft. long; No. 5 mill, which is a Krupp No. 5 size, is 5 × 26 ft. The remainder of the Krupps and Nos. 10, 11 and 12 are equal to Krupp No. 4, 5 × 21 ft.

It will first be noticed that, with rare exception, all mills worked practically full time. It will next be noted that the power varies considerably. This is particularly seen in the last three mills, with the bevel-gear drive. Doubtless this drive contributes decidedly to the extra power consumed, as does also the heating of the roller axles supporting the tire end, through insufficient area. In addition these mills are well loaded with crushing rock, working on the reground sand. It should also be noted that the ribs in the El Oro lining lift the pebbles high.

TABLE XII.—POWER CONSUMPTION OF TUBE MILLS, EL ORO MINING CO.

Mill number	Type	Rated horsepower of motor drive	Actual kilowatts	Hours	Kilowatt-hours	Corrected line load	Cost
1	Abbé.......	50	35	737	25,795	26,886	$201
2	Abbé.......	50	35	707	24,745	25,790	193
3	Krupp......	....	30	732	21,960	22,890	171
4	Krupp......	....	40	651	26,040	27,150	203
5	Krupp......	....	67.5	711	47,992	50,000	374
6	Krupp......	75	46	707	32,522	33,900	254
7	Krupp......	75	42	732	30,744	32,040	239
8	Krupp......	75	44	592	26,048	27,150	203
9	Krupp......	75	42	720	30,240	31,520	236
10	C. & W.[1]....	....	73	730	53,272	55,740	414
11	C. & W.[1]....	....	72	724	52,160	54,370	406
12	C. & W.[1]....	....	79	731	57,696	60,136	450
4, 5, 6	Shafting....	....	24	735	17,640	18,390	137
....	....	....	....	....	....	....	$3481

[1] Chalmers & Williams bevel-geared.

All these motor-driven mills are belt connected. At the Mexico and Esperanza mills, the drive is direct through a flexible leather-link coupling, which is preferable. The mills are continually fed by hand with regrinding rock, introduced through the lower end by means of a worm feed, so that the load is fairly constant.

Power at El Oro, supplied by the Mexico Light & Power Co. from Necaxa, 175 miles distant, costs $50 per horsepower-year. This low rate permits the extensive use of tube mills, which, as shown, are large consumers of power. Where power cost is high, through unfavorable local conditions, the extensive use of tube mills is questionable. There is considerable difference in the performance of these mills. I have had experience with tube mills of diverse makes, and my own preference is unqualifiedly given to the Krupp mills.

For machines of such simple construction, it is surprising what inherent defects rapidly manifest themselves in both inferior design and workmanship. The heavy weight of the mills, lining, pebbles and charge, with the rapid rotation, from 27 to 32 r.p.m., are doubtless integral factors in testing both design and construction, affording, at the same time, explanation of the large power consumption.

Smooth Lining for Tube Mills (By John Tyssowski).—Of late there has been much discussion as to the best lining for tube mills, and the old silex liners have in many cases been replaced by the El Oro and similar types. The displacement of the silex lining is mostly due to the fact that the other types of lining can be renewed much more quickly and are cheaper. It is interesting to learn the success of the smooth lining at the Butters', Virginia City, Nev., plant. Here 20 stamps, crushing to 2½ mesh supply the ore for four tube mills. The tubes are 5 × 22 ft. and are run 36 r.p.m. The linings are made up of rectangular pieces of smooth white iron 1¼ in. thick, 8 in. wide and 48 in. long. These iron plates are set in the tubes edge to edge and are bolted to the shell of the tube mills with ⅞-in. plow bolts. It has been found that such linings, grinding the coarse, 2½-mesh material, last about nine months on Virginia City ores. The El Oro linings will do little, if any better and are more expensive. The duty of the mills is about the same with each type of lining, and either can be renewed in about the same length of time. In the Butters' mill the El Oro lining was tried first, and later discarded for the smooth plate lining which is now used exclusively.

Tube-mill Linings.—Rand tube-mill operators have not as yet settled on the best type of lining, and some curious theories have been advanced in discussions of the several types. It is generally admitted elsewhere that the object should be to prevent the slipping of the pebbles on the lining as much as possible. Yet some operators in Africa advocate the silex because "by far the greater bulk of the grinding done in a tube mill is done on the liner of the tube," a statement which might meet with vigorous denial from operators in Mexico. Based on the quoted statement, the ribbed lining is not so effective, and the fact that more power is required to run a mill with ribbed lining is considered a disadvantage. The average life of the silex lining which is used on the Rand is given as 85 days, and the time required for replacing a lining as 24 hr. The ribbed lining lasts from 20 to 24 months. The "peg" lining, made by embedding short lengths of drill steel or round iron in a shell lining of cement, is being experimented with, though few mines could provide the necessary quantity of short drill steel without buying it specially.

Tube-mill Linings in Use on the Rand.—A tube-mill lining invented by Mr. Osborne, late cyanide manager of the Glen Deep, Ltd., has come largely into use on the Rand. It appears to be a modification of the origi-

nal El Oro lining. It consists essentially of two bars placed in such a manner that with the aid of cement concrete they lock themselves in the interior of the tube mill. The horizontal bar is about 2 in. long and about ¾ in. wide. The other bar, at right angles to the first, is practically of the same section as a grizzly bar, and is 4 in. long and ¾ in. in thickness at the end projecting into the mill, and 1½ in. in thickness at the base. Concrete is laid on the bottom and part of the sides of the tube mill, with the bars placed in position, as shown in Fig. 50. The bars should be kept in position with wood framing and wedges, while the other portions of the lining are being laid. Neglect of the precaution to hold the bars in position by framing until all the bars are laid and the concrete

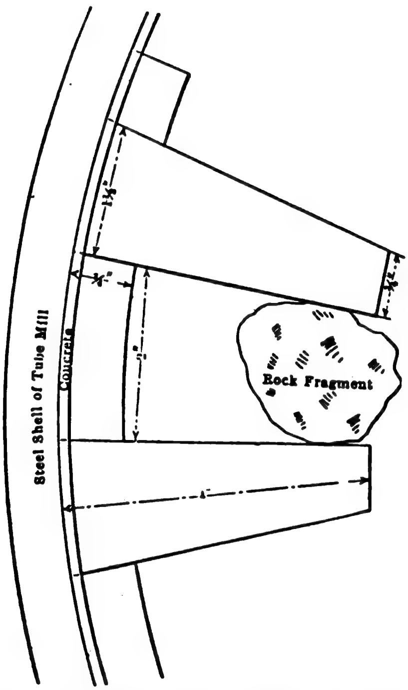


FIG. 50.—SECTION OF TUBE-MILL LINING.

is set recently led to a fatal accident; a native was killed by the lining collapsing upon him. This lining holds banket fragments between the upright bars, which are made of specially hardened steel, and last without renewing for about a year, when the bars need replacing. The use of banket for pebbles is now universal and at the Dreifontein mill of 220 stamps, about 1200 tons per month of banket are thus crushed direct in the tube mills.

White-Schmidt Tube-mill Lining.—A new tube-mill lining has been

suggested by W. A. White and W. F. Schmidt, of the East Rand Proprietary mine, Transvaal, U. S. patent No. 1,068,289. A preliminary shell of steel is laid, composed of thin, narrow bars laid edge-to-edge and cemented together. Upon this lining are placed blocks for receiving the rest of the lining. This, as shown in Fig. 51, is composed of thin, wide bars of steel, set in chocks and arranged radially. Be-

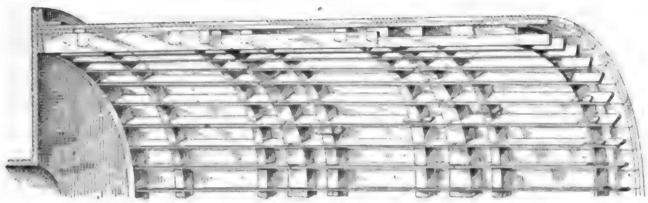


FIG. 51.—WHITE-SCHMIDT TUBE-MILL LINING.

tween the bars is formed a trough into which pebbles are supposed to pack and form a lining essentially like the El Oro lining. The bars may be set away from the shell by resting them on the chocks, the advantage of the system being that the bars may be practically entirely worn out before renewal is necessary.

Convex End Liner for Tube Mill.—With the usual end liner used in tube mills, the function is simply to prevent wear to the greatest possible extent. A new device, however, has been produced by H. C. Holthoff,

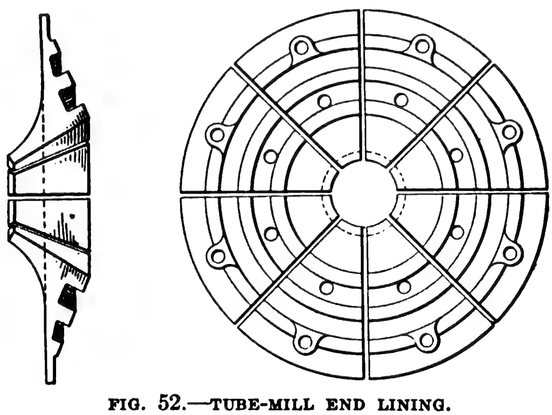


FIG. 52.—TUBE-MILL END LINING.

U. S. patent No. 1,018,320, the object of which is to assist in the grinding operation. To accomplish this object, the liner casting is made convex, as shown in Fig. 52. Grooves or corrugations are cast in the end liner, concentric with the entrance and discharge openings, for the purpose of catching and holding pebbles, thus forming a wear-reducing lining of itself, as is the case with the El Oro lining. End liners of this convex shape

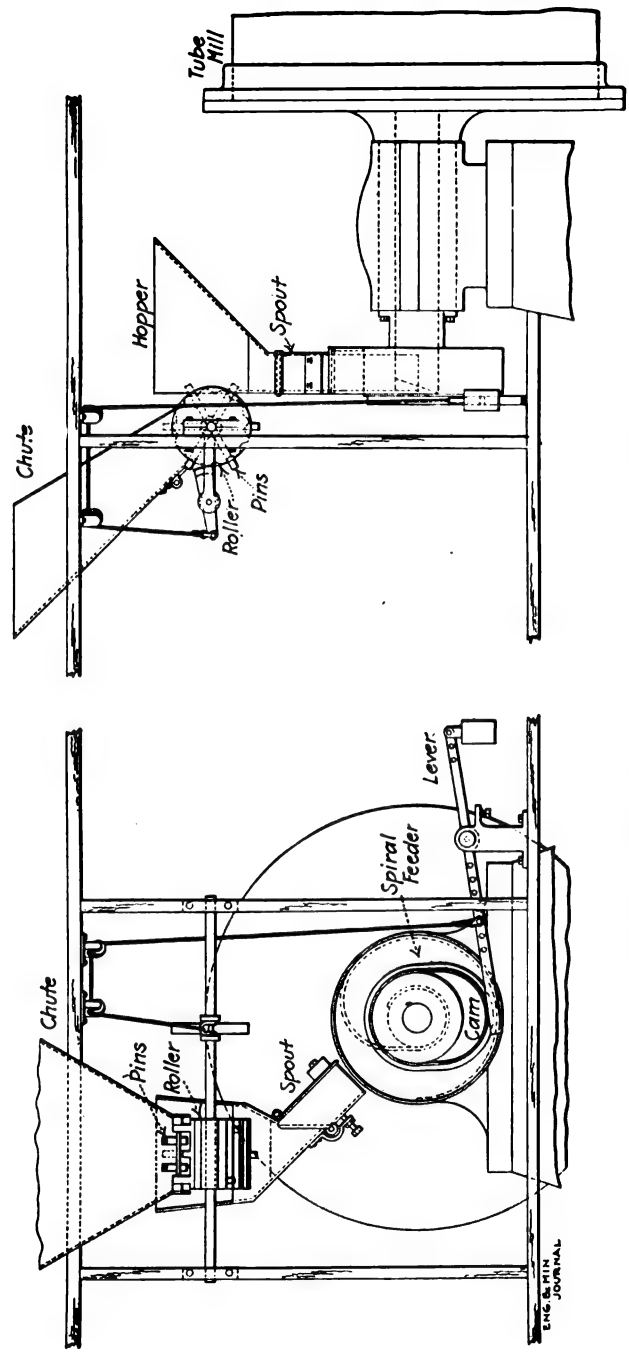


FIG. 53.—PEBBLE-FEEDING DEVICE FOR TUBE MILLS.

are said to prevent the exit of the larger ore particles and pebbles, throwing them back toward the center of the mill where they perform more grinding.

Method for Handling Tube-mill Pebbles.—A simple and convenient method for charging the regular daily quantity of pebbles into the tube mill has been devised for the new Nipissing mill at Cobalt. By this method the pebbles are brought in cars to a convenient point back of and above the tube mills. From this point a wooden launder is laid which reaches to the tube mills and terminates directly over the spiral scoop feeder. A hole is cut in the bottom of the launder at this point, and pebbles dropped through this hole into the feeder at the moment when the opening of the feeder is coming up under the launder. This system is believed to be simpler than any yet suggested and avoids carrying and lifting the pebbles directly into the mill and also the nuisance of having pebbles piled up around the mill, a condition which usually obtains when the pebbles have to be lifted into the mill, either at the feed or discharge point. Pebbles placed in the launder, roll by gravity down to the point over feeders and here they are controlled by the operator who feeds them into the mill at the proper moments when the feeder opening can receive them.

Pebble Feeder for Tube Mills.—With the object of feeding pebbles uniformly and continuously into tube mills, J. E. Thomas, of Germiston, Transvaal, has invented a mechanical means for which he has been granted U. S. patent No. 1,045,342. The device comprises a chute which leads the pebbles up to a hopper from which they are discharged into the spiral feeder of the mill. A fluted or corrugated roller delivers the pebbles from the chute into the hopper supplying the mill, and it is provided with projecting pins which protrude through corresponding slots in the hinged bottom of the chute and prevent choking or jamming of the pebbles in the chute. Movement of the shaft upon which the roller is mounted is provided by a friction wheel on the shaft moved by a rope from a lever actuated by means of a cam or eccentric on the casing of the spiral intake. The movement of the lever can be regulated and so made to control the movement of the fluted roll governing the feeding of the pebbles. The pebbles, after entering the hopper, are conveyed through an inclined spout which delivers them directly above the entrance opening of the spiral feeder, into which they fall and are conveyed inside the tube. The movement is so set that the pebbles will be dropped into the intake of the spiral feeder at the proper point of the revolution. By means of this device the chute can be maintained free of pebbles, which will automatically be fed into the tube mill without especial care on the part of the operator. The feeding being continuous, the replacement by new pebbles more nearly approximates the natural rate of wear and

avoids the fluctuating efficiency caused by allowing great wear to take place and suddenly replacing it at one time. The accompanying drawings, Fig. 53, illustrate the arrangement of the mechanism.

Feeder for Tube Mill (By H. Sharpley).—The feeding device for tube mills, shown in Fig. 54, is so made that it can be reversed by changing the manner of bolting the parts together. The advantages of such a feeder are that it is a simple matter to make the change when it becomes necessary to reverse the direction of drive; it is made in sections that may be readily renewed; pebbles, once in the lifting chamber *L* cannot fall back into *R*, the receiving chamber, and being "built up" it is cheaper to construct than a solid casting. The feeder consists of five parts, *R* is the receiving chamber, *S* the lifting chamber, *C* the connecting flange, and *D* a division plate with inlet hole of suitable area, which area is governed by the sectional area of the trunnion of the mill. The plate *D* is bolted between *R* and *S*, as shown in Fig. 2, in such a position that *I*

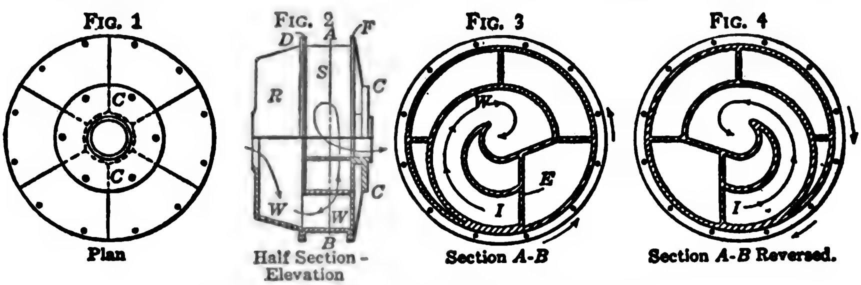


FIG. 54.—A REVERSIBLE TUBE-MILL FEEDER.

comes on the spiral side of the division at *E*, Fig. 3. At *F*, in Fig. 2, is shown a division plate, which is placed between *S* and *C*, and in which there is a hole corresponding in position and diameter with the inner diameter of the tube-mill trunnion. To reverse the feed it is only necessary to loosen the bolts and change *S* from the position shown in Fig. 3 to that shown in Fig. 4, and tighten again the bolts that hold the parts together.

Concentrate Feeder for Tube Mill (By John Tyssowski).—At the North Star, Grass Valley, Calif., concentrates from the stamp mills all go to the Central cyanide plant where they are ground in a tube mill before being added to the pulp for cyanide treatment. The concentrates are brought to the plant in small, V-shaped, side-dump cars and discharged into a circular vat from which they are fed to the tube mill. Owing to the dense mass into which the sulphurets tend to pack, they are difficult to handle and several types of feeder were tried before the one now used was evolved. This apparatus is essentially a disk excavator, suspended over the vat and rotating in it. The blades are set on two arms

and are inclined and spaced so as to push the material uniformly towards the center of the vat as the feeder revolves. The arms of the feeder are suspended by a vertical shaft which is provided with a sliding keyway and driven by a worm gear. The upper end of the shaft is attached to a rope which passes over a couple of pulleys to a counterbalance weight. By regulating this weight and the speed of revolution of the feeder, any desired rate of feed can be maintained. When the vat is being filled with sulphurets, a sectional piece of 6-in pipe is placed over the central discharge gate of the vat. Thus an opening is maintained through the center of the tank. As the feeder lowers, sections of pipe are removed so that the top of the pipe is always kept below the surface of the concentrates in the vat. The material from the feeder is discharged into a launder and washed directly into the tube mill. Little power is needed to operate such a feeder, and the only attention it requires is for taking out sections of the central pipe at infrequent intervals. It is easy to control and insures a constant feed.

The Hardinge Mill

Standard Hardinge Mill Sizes.—The increasing use of the Hardinge conical pebble mill makes interesting the data given in Table XIII, compiled by the Hardinge company. The short cylinder mills are recommended for fine grinding with a minimum of slimes while the longer cylinders are better adapted for grinding with a maximum of slimes. Size of mills is given in diameter of drum in feet and length of cylindrical portion in inches. Capacity and power depend upon weight of pebble charge and will vary according to hardness of material to be ground as well as size of material fed and product required. The lining is made up of silex (flint) blocks, or a combination of ribbed iron or steel plates with Hardinge lifting bars and silex.

TABLE XIII.—STANDARD SIZES OF HARDINGE CONICAL PEBBLE MILL

Size	Floor space	Weight Pounds				Horsepower	Nominal capacity per hour
		Mill	Silex lining	Pebbles			
				Mill charge	Supplied		
Ft. In.	Ft.						
4½×13	6×8	4,000	2,000	1,500	2,000	6–8	1 –1½ tons
[1]4½×72	8×15	8,000	5,000	5,000	6,000	15–18	1½–2½ tons
6 ×16	8×10	7,500	4,000	3,500	4,500	12–15	1½–2½ tons
6 ×22	8×11	8,000	4,500	4,000	5,000	15–18	1½–3 tons
6 ×48	11×12	8,750	6,000	6,000	7,000	20–25	1½–2½ tons
[1]6 ×72	11×16	10,000	7,500	8,000	10,000	25–30	2 –3 tons
8 ×22	11×12	11,000	7,000	8,000	10,000	30–35	2 –3½ tons
8 ×30	11×13	11,500	7,500	10,000	12,000	35–45	2 –4 tons

[1] For very fine or Slime grinding only.

Pebble Lining for Hardinge Mills (By David Cole).—In view of the unsatisfactory situation prevailing at present in the silex market, it is interesting to know that the flint pebbles regularly employed as a grinding medium may also be successfully used for lining pebble mills, and are being so used at the No. 6 Concentrator of the Arizona Copper Co., Ltd., at Morenci, Ariz., in 8 ft. by 22 in. Hardinge mills. In our experience of the past nine months, silex linings have not averaged more than 75 days in use, the longest period being one of 117 days. One of the pebble-lined mills was started in service June 20, 1913 has now been in operation more than 125 consecutive days, and at this writing (Nov. 15, 1913) the lining is in excellent condition. The pebbles are only slightly flattened and apparently not half worn out, thus indicating a life of at least four months, as compared with the above record for 2½-in. Belgian silex blocks.

Lifters are necessary with silex, and are hard to maintain. With the pebble lining, the whole interior of the grinding chamber is rough, thus eliminating the necessity for "lifters." The "cascading" effect is ideal for the reason that the slip of the charge is absent, or limited. The small pebbles of the charge fit themselves into the spaces between the ends of the large pebbles in the lining, and a large percentage of them remain wedged in these spaces for some time, preventing slippage. The lining in the short cone end of the mill becomes flattened first, for the reason that the charge necessarily slips on the steeply inclined surface of that end.

The cost of lining with No. 4 pebbles is about the same as with the regular 2½-in. silex blocks. Mortar made with one part portland cement to 1½ parts clean, sharp sand, is used in the same way as when lining with silex. The larger pebbles are used in the cylindrical part of the shell, in the short cone, and in the larger part of the long cone, and successively smaller ones in the remainder of the long cone. Pebbles are set endwise, tight together. The inlet and discharge throats are lined with small worn pebbles instead of iron lining as formerly. The inlet and discharge funnels are coated with neat cement to a thickness of ¼ in., painted on, in successive layers. Thus, no portion of the shell or the interior of the mill is left unprotected against "copper water." Cement covers the pebbles for two-thirds of their height.

I think no flint need be "scrapped," as the worn pebbles of discarded lining may be used subsequently in the pebble charge. The pebble lining tends to increase capacity through preserving maximum "cascading" effect. No. 5 and No. 6 pebbles will be used in the 8-ft. by 36-in. new Hardinge mills now being installed, and we expect a life of 6 months with these.

Special Designs

Baltic Regrinding Plant, Redridge, Mich. (By A. H. Sawyer).—One of the greatest problems in ore concentration is the treatment of middling

products from jigs and tables. It is here that the greatest losses occur and where the most improvement is possible. In order to effect greater savings from this product an efficient and economical grinding machine is necessary. In the Lake Superior district various types of grinding machines, including chilean and Huntington mills, have been tried, but because of their low capacity and high maintenance cost they have not been successful in solving the problem. With the advent of the low-pressure turbine, producing power at less cost, and the use of tube mills

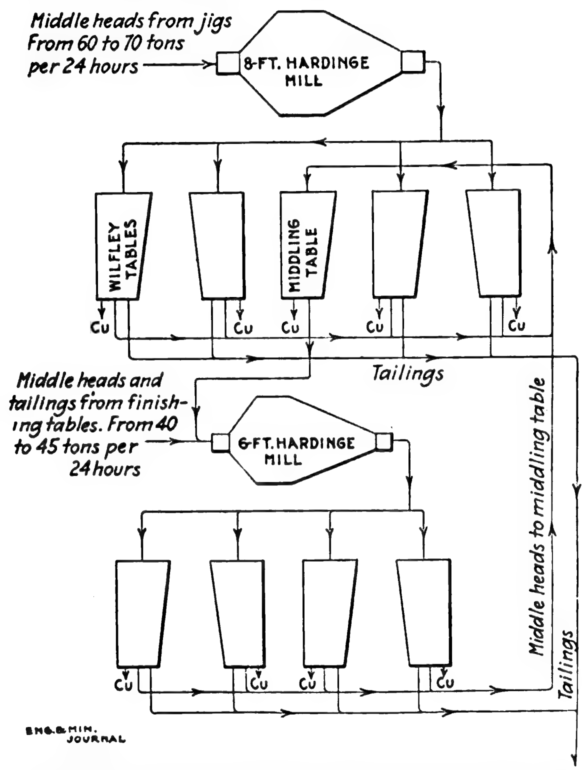


FIG. 55.—FLOWSHEET OF BALTIC REGRINDING PLANT.

mechanically more perfect, the treatment of middlings promises to be much more successful than in the past.

The old section of the Baltic mill at Redridge, Mich., which contains four Nordberg simple steam stamps, is being equipped with a regrinding plant divided into four units, one for each head. The plant is being built in the basement, previously not used, so that no alteration in the mill proper was necessary. Each unit consists of one 8-ft. by 30-in. and one 6-ft. by 22-in. Hardinge mill and nine Wilfley concentrating tables. Feed launders connect adjacent units so that if one head is shut down the

other feeds the mills of the idle head as well as its own. The flow sheet of one unit is shown in Fig. 55. The middlings from the jigs and the

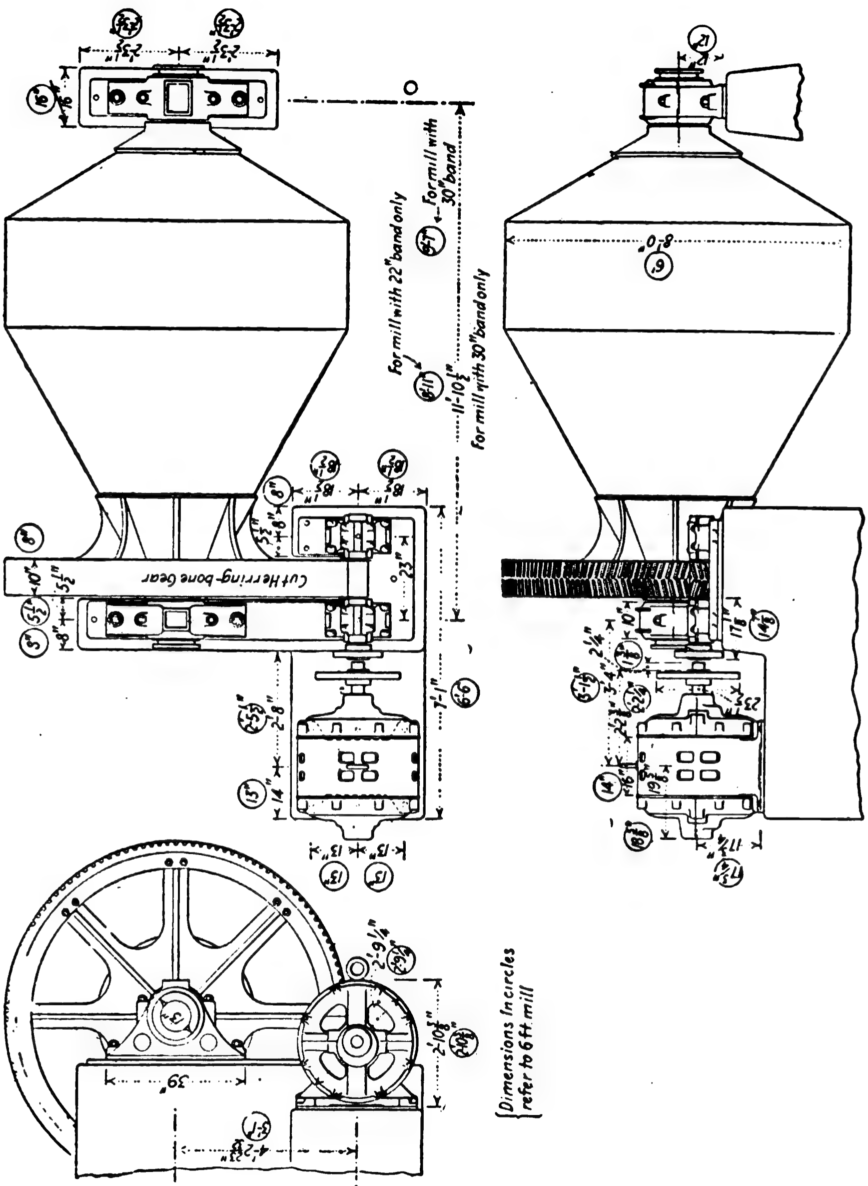


FIG. 56.—DETAILS OF HARDINGE MILLS USED AT BALTIC REGRINDING PLANT.

middlings and tailings from the finishing tables are fed to the mills by gravity. The products from the tables of the regrinding plant, however, have to be elevated and for this purpose centrifugal pumps will be used.

The mills were built in the shops of the Champion Copper Co. at Painesdale under the Hardinge patents, but important changes were made in the mechanical design. The approximate weight of mills, lining and pebbles is as follows: 8-ft. mill, 8000 lb., lining, 9000 lb., load of pebbles, 12,500 lb.; 6-ft. mill, 5600 lb., lining, 6000 lb., load of pebbles, 3800 lb. The mills are fitted with cut herringbone gears and pinions, the gear for the 8-ft. mill being 8 ft. in diameter by a 10-in. face with 243 teeth; for the 6-ft. mill the gear is 6 ft. in diameter and 8-in. face with 213 teeth. The pinions are 5.2 in. and 4.45 in. diameter, respectively, and each has 13 teeth. The ratio of reduction is, therefore, 18.7:1 and 16.4:1. These gears are expected to reduce the power consumption 10% from that required with plain cast gears. The assembled mill is shown in Fig. 56.

The bearings for the mill and pinion rest on sole plates, which are set in concrete foundations. This arrangement allows placing the pinion bearings close to the pinion avoiding the more or less flexible pinion shaft of the regular Hardinge mill. It also results in a more rigid connection between the mill and pinion. The pinion bearings are adjustable by means of setscrews. The axis of the mill is level rather than inclined as in the Hardinge design, and several of the parts are heavier.

The mills will be driven by 50- and 25-hp. motors, respectively, mounted on the same concrete foundations as the mills. These motors are of Westinghouse manufacture, three-phase, 60-cycle, 2200-volt, and run at 500 r.p.m. They are connected with the pinions through flexible rope couplings, and rotate the mills at 26.7 and 30.5 r.p.m., respectively. The tables for the regrinding plant will be driven for the present from the line shafting, but later motor drive will probably be substituted throughout the whole mill. The current for the motors will be generated by the turbogenerator recently installed. This unit is a 1250-kw. low-pressure Allis-Chalmers turbine coupled to a three-phase, 60-cycle, 2300-volt generator running at 1800 revolutions per minute.

Changing a Tube into a Cone Mill.—It is of particular interest that a 7 × 12-ft. tube mill, which was installed at the Morning mill of the Federal Mining & Smelting Co., Idaho, was so changed as to make it into a conical mill. The circumstances were that Hardinge mills were found to do better work than the cylindrical tube mills and they were installed, the tube mill being laid aside. Later, having need for more capacity, the tube mill was reclaimed and made into a cone mill. Transition was effected by bolting heavy timbers inside the tube mill, forming a cone as shown by the dotted lines in Fig. 57. Over the timbers, steel rails were spiked or bolted to form a lining which resembles the El Oro ribbed type. Pebbles pack in the interstices between the rails, as with the El Oro lining, taking up most of the wear.

The efficiency of the mill is shown in Table XIV, table of screen

classification of feed and product both before and after the change. The capacity of the mill before the change was 98.64 tons per day, while afterward it was 88.2 tons, a reduction of about 10.5%. The horse-power required before the change was 75.6, and afterward, 65, a saving of about 14%. It is probable, however, that the power stated as required before the change is underestimated. The revolutions of the mill were not changed, remaining at 22¼ r.p.m. in each case. Moisture in feed before the transition was 58.9%, and after it, 57.7 per cent.

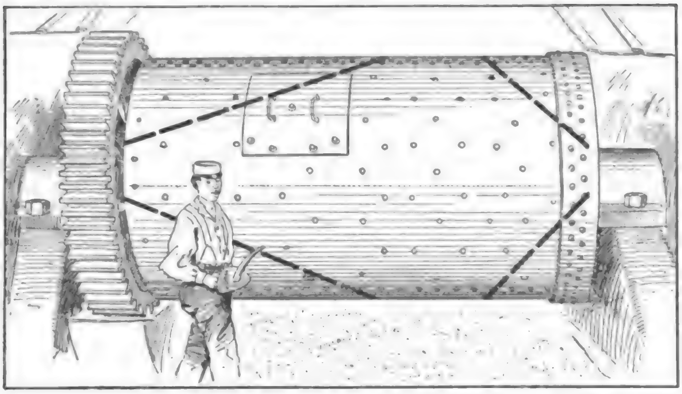


FIG. 57.—TUBE MILL AS CHANGED INTO A CONE.

The change made by transforming this machine into a conical mill would probably infringe the Hardinge rights, but the controllers of the latter issued a special license to cover the case, a procedure which they offer to follow, within certain conditions, wherever mill owners desire to make a similar change.

TABLE XIV.—CLASSIFICATION OF WORK OF TUBE AND CONE MILLS

Screen	Before change Feed, %	Product, %	Screen	After change Feed	Product
+10	29.5	....	+8	19.10	.....
+20	45.5	....	+10	22.05	.....
+30	12.0	0.5	+14	17.05	.....
+40	6.0	0.5	+20	14.50	0.10
+60	4.0	2.5	+28	11.80	0.45
+80	1.5	5.0	+35	7.05	2.25
+100	1.5	8.0	+48	4.20	4.50
+150	....	5.5	+65	1.99	7.65
+200	....	20.5	+100	1.05	10.50
−200	....	57.5	+150	0.60	17.20
.....	....	....	+200	0.20	16.00
.....	....	....	−200	0.35	41.20

The Quinner Dry Pulverizer and Separator.—The machine shown in the accompanying drawing was devised by M. Quinner and is suc-

cessfully used in the Altar district placer region, near El Tiro, Mex. The construction is similar to that of the trommel screen; being cylindrical in form and surfaced by steel bars $\frac{1}{8}$ in. apart. The cylinder is usually about 6 ft. long and 3 ft. in diameter, and revolves not faster than 25 r.p.m. on flanged wheels set under it. A shaft, operating on its own journals, runs through the center of the machine and to it are attached about three dozen chains set spirally and at uniform distances from each other. Iron slugs are welded to the ends of the chains. These are about 6 in. long, square-shaped, about $2\frac{1}{2}$ in. thick, and are placed so that the sharp edges strike the material fed through. Fig. 58 illustrates

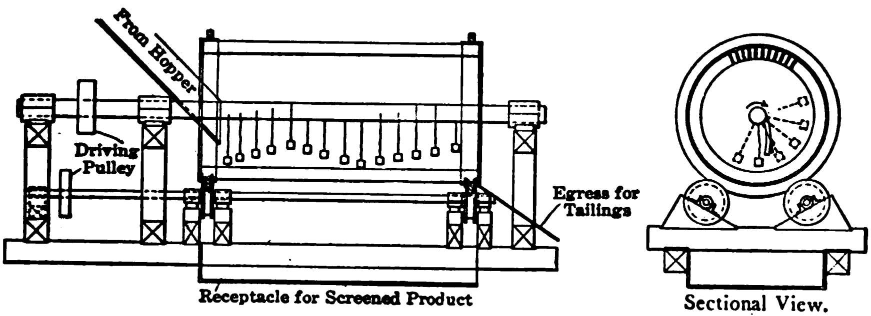


FIG. 58.—QUINNER DRY PULVERIZER AND SEPARATOR.

the construction of the machine. The shaft turns in an opposite direction to that of the trommel, at an approximate speed of 375 r.p.m. The spiral arrangement of the pendants carries the rocks and pebbles through rapidly; the pulverized cement, black sand and gold falling between the bars into a receptacle placed beneath. While the chains are forced to practically a rigid position when running, they rarely break when a substance too hard to shatter is encountered—an advantage over rigid rods. One machine will handle 500 tons of suitable ore per day. The chief elements in favor of this apparatus are that it requires no water, except for steam operation, and discards the bulk of valueless pebbles and rocks without crushing them into the material that is later dry-washed for gold.

III

ORE DRESSING—WASHING, SEPARATING AND CONCENTRATING

PRELIMINARY CLEANING AND SEPARATION

Washing

Iron-ore Washing Calculations (By George C. Olmsted).—The accompanying illustration, Fig. 59, shows a set of curves which may be used in connection with an iron-ore washing plant. Without calculation they show how many tons of raw ore of given iron content, are

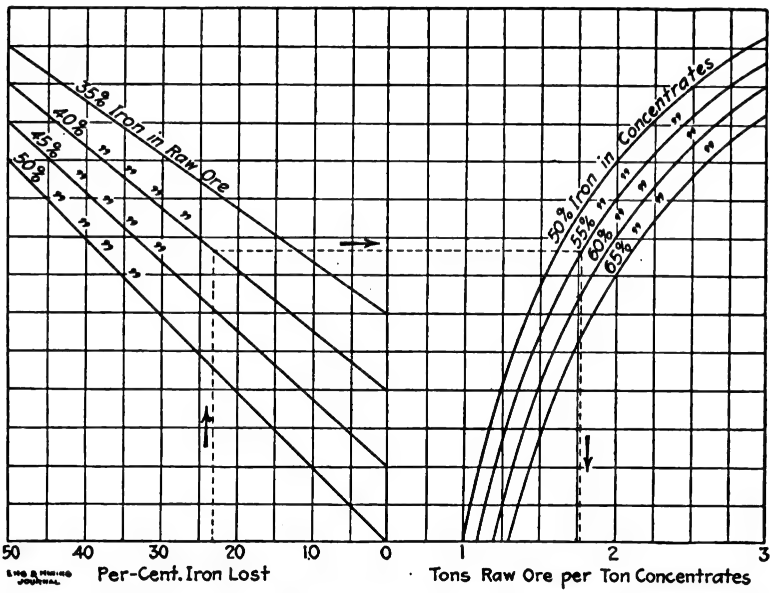


FIG. 59.—IRON-ORE WASHING CURVES.

needed to produce one ton of concentrates of required iron content when a certain amount of the iron is lost. For example, a raw ore containing 40% iron is to be washed so as to produce 55% ore; in the process 23% of the iron is lost. How many tons of raw ore are needed per ton concentrates? Follow up the ordinate from the percentage of iron lost, to the line showing percentage of iron in ore, then across along the

abscissa to the curve giving the percentage of iron in the concentrate. The length of this abscissa (1.78) is the tons of raw ore required.

Removing Chips from Ore.—Chips of wood unavoidably become mixed with the ore from all mines and where the ore is to be milled these chips give rise to considerable annoyance because they choke screens and classifiers. The device illustrated in Fig. 60 was originally designed for removing chips from the coal produced from the mines of the Burnside colliery of the Philadelphia & Reading Coal & Iron Co., states *Coal Age*, but there seems to be no reason why it could not be used as effectively in removing chips from ore. The apparatus consists of a V-shaped tank filled with water, and a double-strand scraper line. Coal or ore is led into the tank by a chute which terminates at the water level. It, of course, settles to the bottom of the tank and is carried up by the flights of the scraper line, being discharged over a perforated screen plate

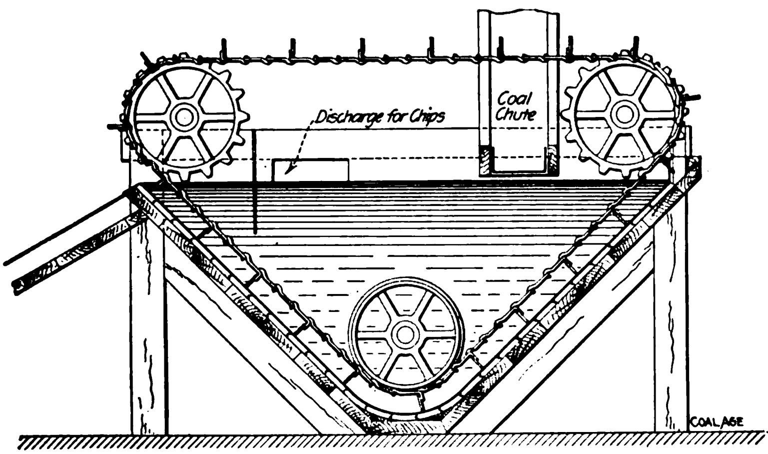


FIG. 60.—APPARATUS FOR REMOVING CHIPS FROM COAL OR ORE.

to remove the small quantity of water which accompanies it. The flights are notched and perforated to prevent their discharging a large amount of water. The chips which are brought in with the ore float on the surface. Water is being constantly fed into the tank and runs off at a rectangular opening in the side, causing a current in that direction, which carries the chips with it. A partition of perforated screen plate is inserted just behind the forward sprockets of the conveyor line and carried down well below the surface of the water. This prevents the pieces of wood from floating over and mingling with the ore at the place where it is discharged. The construction of the machine is simple and inexpensive, the power required is small, and the necessary water can, of course, be used over and over again, a number of times, from the point where it is discharged.

Screening

Improved Method of Holding Grizzly Bars.—There are two methods of securing the bars in a grizzly, of which the less desirable is the one in common use. In the accompanying sketch Fig. 61, from a paper by C. O. Schmitt, read before the South African Association of Engineers, the upper figure shows the common method of holding the bars by rods and distance pieces, while the lower figure illustrates a scheme for holding them independently. The latter method has the advantage that each bar can be taken out and replaced when worn, with a minimum of labor; while in the former case the whole set has to be lifted by block and tackle.

Joplin Trommels.—Two sizes of trommels are used in the Joplin district of Missouri, 48 × 96 in. and 36 × 96 in. The framework of both is the same except as to the diameter of the hood and the length of the arms of the spider. The arms are protected at the ends by castings that

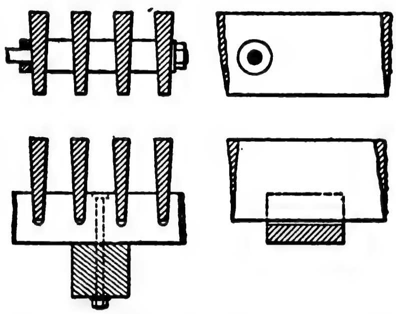


FIG. 61.—METHOD OF HOLDING GRIZZLY BARS.

are fastened to the arms by a ¾-in. rivet. This protecting casting is made of two parts, a plate and the shoe. The plate is a casting with walls ⅝ in. thick that come half over the arms of the spider, which are pieces of 2½ × ½-in. iron, bent at the ends so as to fasten by two ¾-in. rivets to the hoops of the trommel, also made of 2½ × ½-in. iron. The shoe is of the same section as the plate in the shank part, but at the bottom is a projection at right angles, so as to cover the heads of the rivets that fasten the arms to the hoops. This is done to protect the arms from excessive wear. The spiders of the trommel are composite, being made of a hub casting, and the arms, riveted together by ¾-in. rivets. The hub is fastened to the shaft by a key and two setscrews, one of which comes over the key. On the hub are ribs for receiving the arms.

In the trommels put on the market by J. A. Rogers, the sockets for the drawband bolt are made by punching two slots through the drawbands to receive a U-clip 2½ in. wide and of No. 8 sheet gage. This clip goes over

the drawbolt and clinches over backward, so that it cannot draw out of the slots as tension is put on the band. Moreover, as the drawbolt is tight against the band, this clip does not have the tendency to turn over as in those having cast-iron sockets for the drawbolts riveted to them.

The ore is fed into a cast-iron hood on the trommel. This hood is of the same diameter as the trommel and its walls are cast ½ in. thick. The side wall comes up 3½ in., and the wearing wall extends out from the side wall 10 in., so that the feed as it comes into the trommel drops on the hood casting which takes the wear instead of the trommel screen.

Trommel for Coarse Dry Ore.—Experience at the mill of the Doe Run Lead Co., operating in southeastern Missouri, showed the necessity of using a specially designed screening trommel when handling undersize from 1-in. trommels. It was found that the impact of the falling ore produced excessive wear on the front legs and first section of the screening trommel. Details of the trommel designed by H. R. Wahl, mechanical engineer for the Doe Run Company, are given in Fig. 62. The trommel is provided with a diaphragm hood with wearing plates. The feed hopper is carried on a pedestal; around this and making a loose fit is the front head of the trommel hood. This annular front head, formed by bolting together four castings that overlap one another at the joints, is in turn fastened by bolts to a flanged separator ring or shell that meshes with a groove in the side of the diaphragm to which it is bolted. This diaphragm has reinforcing ribs raised on the back. The material between these ribs is cut away near the rim or flange to provide the holes *H*, through which the ore feeds to the trommel screen. Between the two heads of the hood are placed wearing plates held in place by square wings that fit into recesses in both heads of the trommel hood. These wearing plates have a slope in a direction opposite to that of the rotation of the trommel so that the ore as it feeds from the hopper or feed chute strikes the wearing shoes and slides gently down upon the shell of the trommel hood; in this way the life of the separator shell as well as that of the diaphragm or back head of the hood is greatly lengthened.

The only drawback to this form of hood is that the wearing plates are somewhat difficult to replace and the ore, if damp, tends to stick to them. On that account their use has practically been abandoned. It might be possible to arrange the wearing plates so that they would be held in raised grooves on the two heads, by bolts, and so could be replaced by removing the holding bolts instead of taking off the front head. They could be designed so as not to come quite so near the bottom. The idea of the wearing plates seems good, as the ore is made to fall gently upon the bottom of the trommel, and excessive wear on the legs of the trommel spider as well as on the first section of screen is avoided. The hood diaphragm is also of service in preventing the scattering of dust through the mill. It

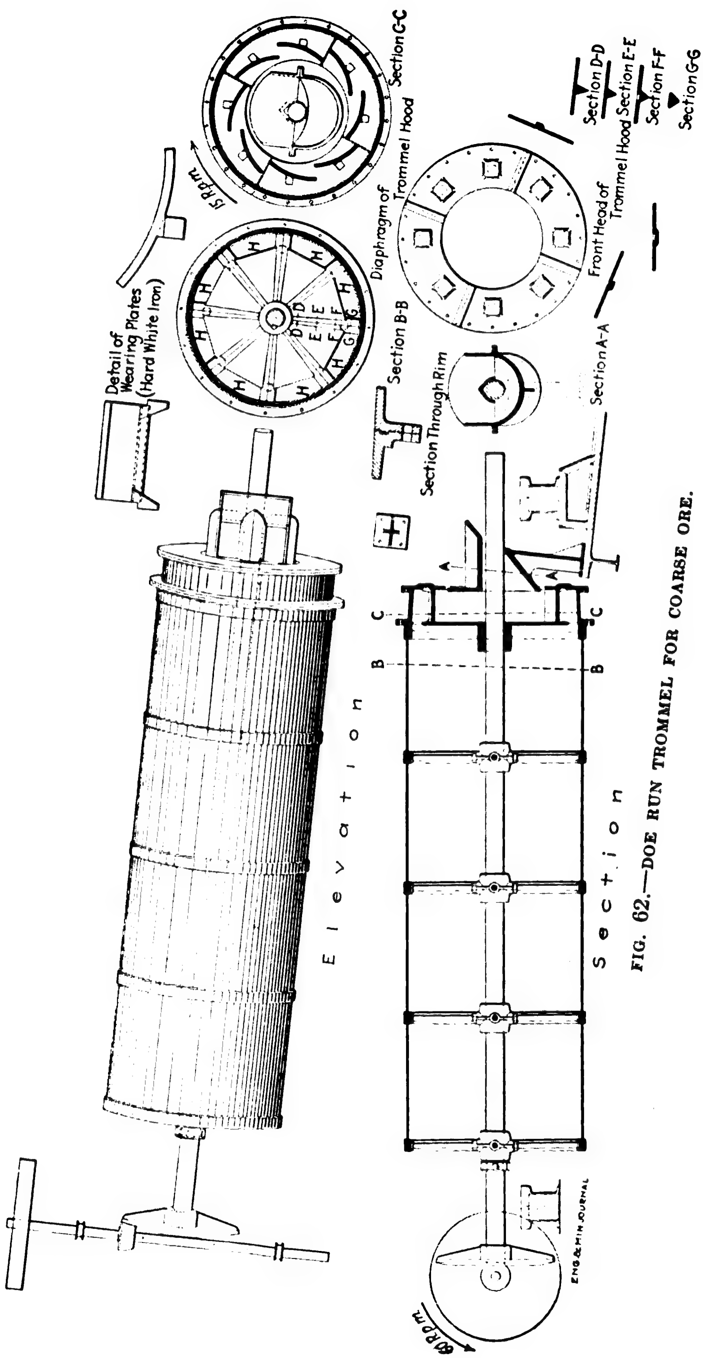


FIG. 62.—DOE RUN TROMMEL FOR COARSE ORE.

has been found that the trommel is somewhat longer than necessary, as practically all the undersize is screened out on the first two sections.

Escape of dust from the trommel through the perforations is prevented by carrying up from the floor on which the trommel is placed, a frame of 2 × 4-in. timbers with a casing of 1-in. planks. Over the top is placed a covering of canvas which is supported upon a framework of six 1¾ × ⅜-in. iron bands that spring from the side frames and are tied together with a wooden center rib and six straps of 1¾ × ⅜-in. iron as shown in Fig. 63. The two center bands are close together owing to the fact that the canvas covering is lapped at that point. This covering is held down on the canopy frame by ¾-in. rods that run through hems; one along each of the bottom edges. This makes it easy to lift the covering and inspect the trommel at any time while escape of dust is prevented. Manganese-steel screens with openings 9 mm. in diameter are being tried on this trommel. They have been used only a few months so it is impossible to state what service they will give.

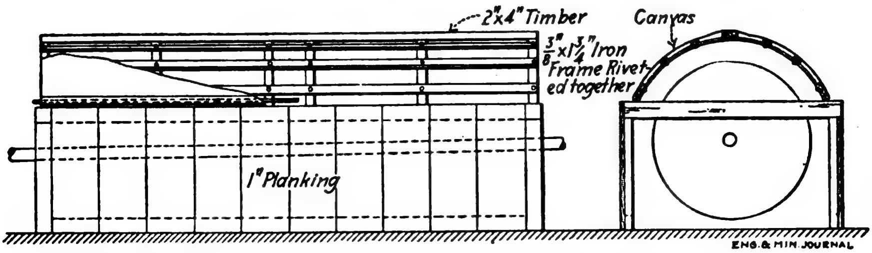


FIG. 63.—HOOD USED ON DOE RUN TROMMEL.

Hexagonal Trommel for Fine Pulp.—At the mill of the Desloge Consolidated Lead Co., in southeastern Missouri, comparative tests between conical and hexagonal trommels proved most instructive. The duty of the apparatus used in the test was the division of the undersize of a 10-mm. trommel into table and jig feed by passing it over a 1-mm. screen.

The conical trommel was 96 in. long with diameters of 36 and 48 in. Four interior rings ⅝ in. thick and 2½ in. wide carried the screen, which was set with a fall of ¾ in. to the foot. Each ring was stiffened with six spokes. The hexagonal trommel was 96 in. long and the circumscribing circles had diameters of 36 and 48 in. There were four hubs on the carrying shaft, from each of which six spokes radiated. Longitudinal, 120 deg. angles, fastened to one spoke of each of the four sets, formed the corners of the hexagon and to these angles were fastened the screen plates. The six screen plates had no internal support other than the angles, except at the ends of the trommel, where 1½ × 1¼-in. straps reinforced the screen inside and out to stiffen its edges, as shown in Fig. 64. There was a 14-in. fall

when the pulp dropped from one face of the hexagon to the other. The screen used on both trommels had 1-mm. round holes, staggered 2½-mm. centers, punched in No. 22-gage steel plates. An outside spray helped to prevent binding in both cases. The character of the rock and the scouring and pounding action of the large volume of oversize prevented much trouble from binding. Both trommels were fed at the rate of 150 tons per 24 hr. and operated at a speed of 20 revolutions per minute.

The tests were carried on for several weeks, so that they show average conditions, and all conditions were as near the same for both trommels as it was possible to keep them. The results of the tests are as follows: Oversize, finer than 1 mm.; conical trommel, 7.02%; hexagonal trommel, 4.77%; life of jacket, conical trommel, 7.1 days, 1065 tons; hexagonal trommel, 12.85 days, 1927 tons. The fact that there was less binding and thus less undersize in the discharge from the hexagonal screen is accounted

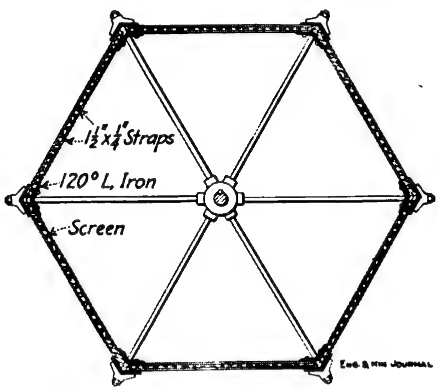


FIG. 64.—SECTION OF HEXAGONAL TROMMEL.

for by the greater agitation which the pulp gets in this screen. The greater life of the screen on the hexagonal trommel is due to the obstructions which the interior rings of the conical trommel offered to the passage of the pulp. This slight obstruction with the low slope used and the greater weight of pulp that accumulated just above the rings resulted in the screens giving away at those points, generally before they were nearly worn out elsewhere. When the interior rings of the conical trommel were made thinner, the life of the screen was increased, but the life of the trommel itself was considerably shortened. It is also well to note that the whole support to a trommel screen should be against outward movement, owing to centrifugal force, rather than against inward distortion. Thus it seems that, for fine screening, the middle and bottom rows of spokes of a conical trommel should have only capping segments to grip firmly and support the screens for a few inches on the inside.

The great advantage of a polygonal trommel for fine screening is that

the surface is in sections. It is difficult to patch a hole in a fine screen and with the conical trommel the entire screen has to be discarded, whereas, in a hexagonal or octagonal screen one section is replaced at a time. When the screens are attached between angle ribs rather than on a frame, the labor and cost of changing are also much reduced.

The swish of the pulp that results from the greater lift in the polygonal-shaped screens causes the ore to be passed more rapidly and results in a cleaner discharge than can be obtained with the conical trommel. When the fall that can be allowed in screening is limited, the greater speed at which the pulp will pass through a hexagonal screen, as compared with a conical one, may be the determining factor in favor of its selection.

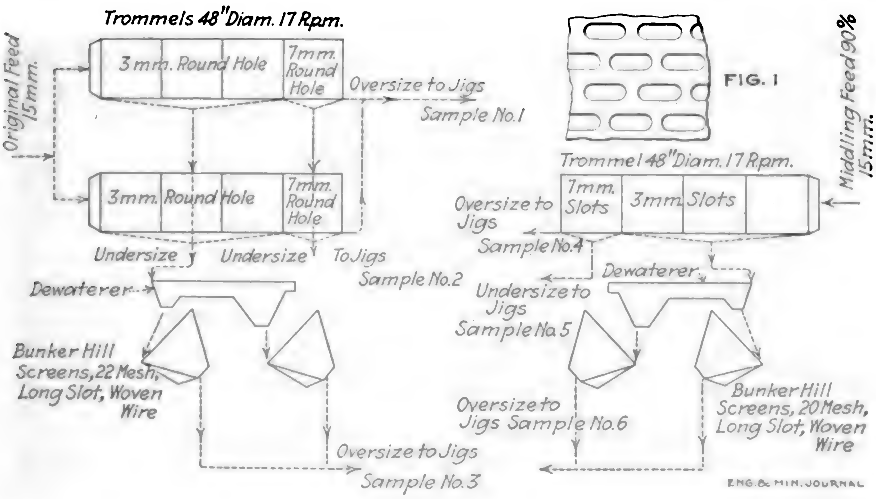


FIG. 65.—ARRANGEMENT OF TROMMELS FOR COMPARATIVE TESTS.

Slotted vs. Round-hole Trommel Screens (By R. S. Handy).—At intervals during the last year I have been experimenting with slotted punched-steel screens in an effort to find something to replace the round-hole screens on trommels, which, in my experience, have always shown low efficiencies. My first order for slotted screens was as follows: One 30 × 151-in. steel screen, $\frac{1}{4}$-in. gage, punched with slots 10 mm. wide by 1 in. long; one 30 × 151-in. steel screen, $\frac{3}{16}$-in. gage, punched with slots 7 mm. × $\frac{3}{4}$-in.; one 30 × 151-in. steel screen, $\frac{1}{8}$-in. gage, punched with slots 5 mm. × $\frac{1}{2}$-in.; slots to run with the circumference and to be staggered, as in the upper right-hand corner of Fig. 65. When these screens were put in the place of round-hole screens, with the width of the slot corresponding to the respective diameter of the round-hole, the jigs receiving their feed from the slotted trommels were swamped with feed and the slotted screens had to be removed. The 7-mm. slotted screen was then put

in place of a 10-mm. round-hole screen. Table XV shows the screen tests on the oversize of these two screens. The 10-mm. slotted screen was substituted for an 18-mm. round-hole, and the 5-mm. for a 7-mm. round-hole screen; each gave as good service and slightly better oversize screening tests than the round-hole screen which it replaced.

Table XV.—Comparative Grading Analyses of Trommel Oversize

	Oversize 10 mm. round-hole, %	Oversize 7 mm. slotted screen, %
+ 10 mm.	84.2	87.4
+ 7 mm.	11.2	11.8
+ 5 mm.	4.1	0.8
− 5 mm.	0.5	....

The results of these tests were encouraging but, with the exception of the 5-mm. size, the slotted screens "blinded" greatly, and there was too large a proportion of elongated pieces in the undersize. To overcome these faults the gage of the steel and the length of the slots were changed and the following screens were ordered: Three 30×153-in. steel screens, No. 14 W. & M. gage, closely punched with oblong slots, running with the circumference, 3-mm. wide by 3/16 in. long; one 30×153-in. steel screen, 1/8-in. gage, punched as closely as possible with oblong slots, as above, 7 mm. wide by 3/8 in. long; screens to be rolled to a 4-ft. circle, with the small side of the slot inside. These screens were lately put into service on the middlings side of the Bunker Hill & Sullivan West Mill No. 2 in competition with double the number of equivalent round-hole screens handling the same tonnage of about the same size material on the original-feed side of the same mill.

Results of a test covering eight hours of regular service of these trommels are shown in Tables XVI and XVII. In Table XVI are given the screen tests on products (samples 1 and 2) from two trommels, each consisting of 180 in. of 3-mm. round-hole screen and 60 in. of 7 mm. round-hole screen; and on the oversize (sample 3) of two Bunker Hill screens with 22-mesh, long-slot, woven-wire screens. Table XVII shows the screen tests of products (samples 4 and 5) from one trommel having 90 in. of 3-mm. slotted screen and 30 in. of 7-mm. slotted screen; and of the oversize (sample 6) of two Bunker Hill screens with 20-mesh, long-slot, woven-wire screens. Reference to the accompanying flow sheet, Fig. 65, will make clear the arrangement of the trommels. It is interesting to note that the slotted screens are showing no tendency to blind, while the round-hole screens have to be cleaned at intervals.

The specifications for the original-feed trommels are as follows: Six 30×153-in. steel screens No. 14 W. & M. gage, closely punched with 3-mm. round holes; two 30×151-in. steel screens, 1/4-in. gage, punched with 7-mm. round holes, 7/16-in. centers; the screens to be rolled as above.

TABLE XVI.—GRADING TESTS ON PRODUCTS OF ROUND-HOLE TROMMELS AND BUNKER HILL SCREENS

Standard screens	Sample No. 1 oversize 7 mm., %	Sample No. 2 oversize 3 mm., %	Sample No. 3 oversize 22 mesh, %
+ 7 mm.	64.6	....	....
− 7 mm.	35.1	....	....
+ 3 mm.	....	89.2	....
− 3 mm.	....	10.8	....
+ 20 mesh	....	....	77.5
− 20 mesh	....	....	22.3

TABLE XVII.—GRADING TESTS ON PRODUCTS OF SLOTTED SCREENS

Standard screens	Sample No. 4 oversize 7 mm., %	Sample No. 5 oversize 3 mm., %	Sample No. 6 oversize 20 mesh, %
+ 7 mm.	89.6	....	....
− 7 mm.	10.4	....	....
+ 3 mm.	....	97.7	....
− 3 mm.	....	2.3	....
+ 20 mesh	....	....	90.9
− 20 mesh	....	....	9.1

The inefficiency of round-hole trommels is due, I believe, to the fact that in their operation they hold the mass of the pulp away from the full opening of the orifices in the bottom of the trommel and up on the side at an angle of about 20 deg. with the horizontal. A vertical projection of a 30-mm. round opening in a screen ½ in. thick on a 4-ft. trommel at this angle is only about 16 mm. wide. That is, the largest particle which could fall freely through a 30-mm. hole at this angle would be 16 mm. wide. This ratio decreases as the sizes grow smaller, until a projection of a 3-mm. hole in a 14-gage screen is little more than one millimeter wide. It is not surprising, then, to find the efficiency of round-hole trommels decreasing as the sizes grow smaller. It was this condition which led me to the slotted form of screen and I endeavored to arrange the length of the slots and the gage of the steel so that a vertical projection of the slot at an angle of 20 deg. would be a circle of the same diameter as the width of the slot. The Bunker Hill ore breaks to a granular form, and in this there is some advantage in favor of the slotted screen, but I believe the dimensions of the slots and the gage of the steel could be arranged to give high efficiencies with trommels on almost any ore.

Hand Sorting

Hand Sorting of Ore.—At a number of the milling plants in southeast Missouri a certain amount of hand picking of the ore is done. The

scheme is as follows: The ore is hoisted from the mine in skips and emptied directly upon a metal-link belt conveyor, approximately 30 ft. long. The belt travels 25 to 40 ft. per min. Two men are usually employed at the picking belt. The amount of waste rock varies considerably. In one case about two 50-ton cars of waste material are picked out per week, at a shaft where the output is five to seven cars per day. In this way at least 7% of the mine output is disposed of at a cost of 20 to 30c. per ton, which is cheaper than passing the barren rock through the mill. The picking floor is usually placed directly over the railroad track and when cars are obtainable the waste rock is tossed into the car. Otherwise it is placed on the floor, which will hold at least a carload, and is shipped out when cars are at the disposal of the mining company. From the picking belt the ore passes over 3-in. grizzlies. That which passes through goes to a storage bin, while the coarse rock goes through a crusher and thence to the same bin. From here it is delivered to the mill by railroad cars.

Crushing without Sorting at Knights Deep.—The decision to dispense with sorting at the Knights Deep on the Rand revives interest in a subject which has received a great deal of thought and attention, says the *South African Mining Journal*, Nov. 30, 1912. The problem the Knights Deep technical advisers and management had to consider was whether it really pays on a large low-grade mine crushing 100,000 tons or more per month to sort at all. The conclusion reached is that sorting at the Knights Deep is unprofitable. Sorting at the surface never was extensively carried out at the Knights Deep. Underground, large quantities of rock were used for waste packs, but on account of this, and because of the fairly large width of the reefs, there never was much scope for close sorting on surface belts or tables. In the Simmer East, now operated by this company, the main gold carrier is a thin and rich grit, and there is the fear that any extensive rejection of waste rock at surface on ore of this nature might result in the native sorters rejecting the real gold carrier of the property. While these considerations, of course, weighed in the examination of the sorting problem, the main reason for dispensing with the practice of rejecting waste was the conclusion that, whereas the amount of waste likely to be rejected at the Knights Deep (say, 3000 tons per month) can be crushed for 7d. per ton, or less than £90, the same quantity would cost £300 per month to eliminate on the picking belts. The average value of the Knights Deep sorted waste is about 1 dwt. and the maximum recovery that can be expected from this waste rock is about 0.7 dwt. Were the mine a high-grade undertaking, it would not pay to burden the mill with ore of extremely low grade, but at the Knights Deep, with its 4-dwt. recovery and large plant, it is easily conceivable that the rejection of waste is an unprofitable operation. In general the

practice of sorting must, of course, be considered in relation to conditions prevailing at each individual mine.

Desloge Picking Shaker.—At the different shafts of the Desloge Consolidated Lead Co. in southeastern Missouri, the ore after passing over a grizzly falls on large shaking apron called the shaker or picker. Here several men with sledge hammers break the large boulders so that they will go into the crushers easily, and also determine whether the boulders are ore or waste. Owing to the muddy condition of the rock it is difficult to tell whether it is ore or waste until broken. It is thought better to do this sorting at the surface, where the men have better light and the sorted waste is at hand for easy inspection and sampling, than to sort underground where the work would be more scattered and where a large part of the fines from the sledging would be lost on the floor of the workings. The ore is a mineralization of a fairly well bedded dolomite and it breaks readily.

The shakers are 10 ft. wide and from 22 to 29 ft. long. A heavy load accumulates on them when the ore is being hoisted fast; occasionally as much as 15 tons of ore are on the shaker decks at one time. The ore is fed along by means of a shake given by a simple eccentric drive. A differential stroke was tried but this was found to throw considerable strain on the drive wheel of the eccentric shaft. The stroke of the driving eccentric is about $4\frac{1}{2}$ in. The shakers are given a slope of from $\frac{3}{4}$ to 1 in. per foot of length; this aids in carrying along the ore. The speed of discharge is regulated mainly by the speed at which the table is moved back and forth. With the shorter shakers a speed of 78 r.p.m. is used on the eccentric shaft, while with the longer ones the speed is 89 revolutions per minute.

The deck is made up of 2-in. planks laid longitudinally on cross pieces of 6×8-in. timber. A covering of $\frac{3}{8}$-in. plate is fastened down on top of the deck. This plate lasts about a year as the places subject to excessive wear are protected by a covering of rails.

The shaker at the No. 3 shaft is suspended from overhead beams by means of old hoisting cables, about 5 ft. apart, which are fastened by shackles and clevises to eye-bolts in the cross timbers and the carrying beams. Rods were tried instead of the cables but were found to be too rigid. The driving shaft, $3\frac{1}{2}$ in. in diameter, carries the two eccentrics which operate the shaker through simple connecting rods. The eccentrics are about 3 in. wide; the outer strap or yoke has an outside diameter of 15 in. and an inside diameter of $10\frac{1}{2}$ inches.

In order to absorb the shock that would otherwise be transmitted directly to the driving belt when a car is dumped on the shaker, the picker should be driven by a heavy, webbed driving pulley that will also act as a flywheel. The webbed driving pulley used on the 10×29-ft. shaker at

No. 3 Desloge shaft has a diameter of 5 ft. and a face width of 10 in. The web between the four spokes is 3 in. thick; the rim 1⅝ in. thick and the spokes which come out to the full width of the face are 2 in. thick. Those spokes, however, that serve as the splice faces for the two parts of this split drive wheel, are 3 in. thick when bolted together.

The shaker feeds the ore to one or two breakers. Brennan multiple-jaw breakers as well as Farrell-Blake crushers are used. The Brennan breakers have been in use for years and are popular because of the light weight of the different parts which allows repairs to be made more easily. While the shaker can be fed from the side, it is preferable to dump the ore on it from the end as it is then spread evenly across the deck, making it easier to sort out waste; the initial velocity of the falling ore also sends

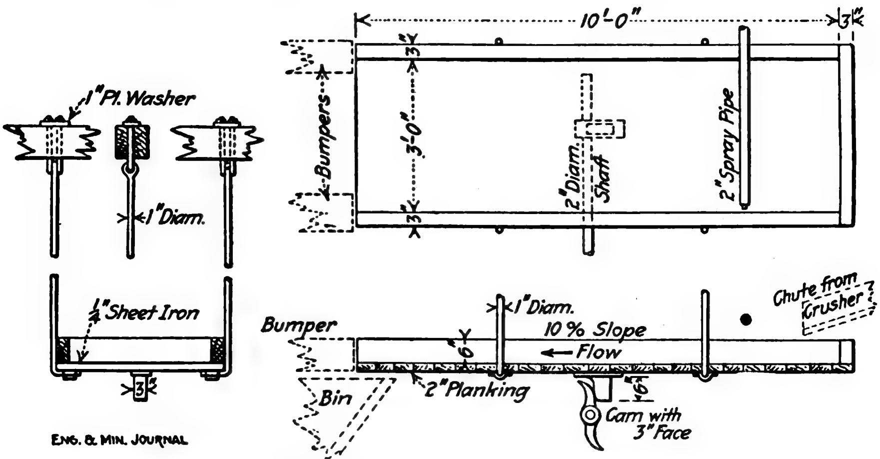


FIG. 66.—BUMPING PICKING TABLE USED IN COBALT DISTRICT.

it ahead in the direction of the feed of the shaker. At the end of the deck are plank chutes for guiding the ore to the mouth or mouths of the crushing equipment. A gate is provided for shutting off the feed to the crushers and when the crusher is heavily loaded, this gate which slants forward toward the head end of the shaker, can be used as a skimmer to regulate the amount of ore being fed to the crushers. Of course such a shaker would be convenient only where there is coarse sorting and sledging to be done. It takes about 6 hp. to run the smaller size of Desloge shaker.

Bumping Picking Table (By Edward H. Orser).—During the first days of mining in and around the Cobalt district, most of the mining companies erected small rock houses in order to cut out as much of the waste rock from the hoisted material as possible. In these plants hand-picking was generally resorted to, and was carried on by either dry picking

or "cribbing," or else on moving tables where the broken ore was subjected to the washing action of a stream of water. In this way the ore was made more easily discernible.

The construction of one of the tables is shown in Fig. 66. It is essentially an inclined tray hung from the roof timbers by four iron rods. The upper end is under the chute from the crusher, and the lower end extends over the bin for waste. The bumping is accomplished by the use of a cam-and-tappet arrangement on the under side of the table and driven from the main lineshaft by pulley and belt. The swing of the table is about 1 ft. On its return it strikes a bumper of timber at the lower end. This action, and the inclination of the tables, which is about 10%,

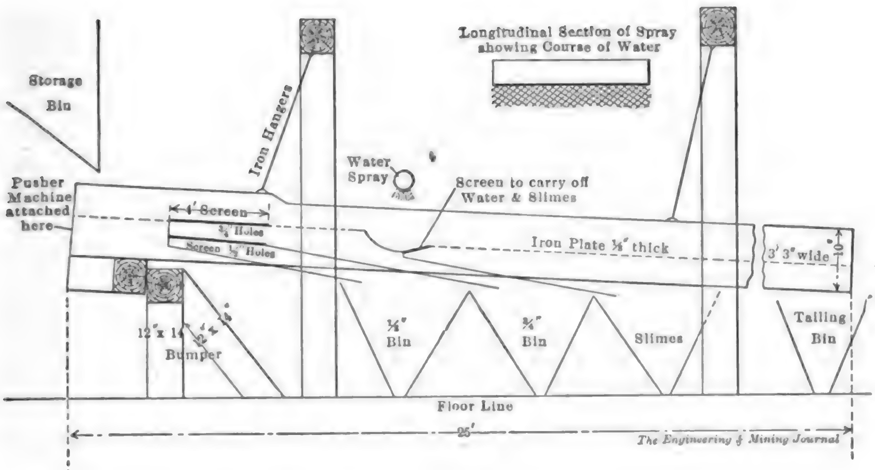


FIG. 67.—SIDE VIEW OF SORTING TABLE AT COBALT.

effect the forward movement of the material at a rate not too fast for clean picking. A 2-in. spray pipe at the upper end of the table supplies the wash water, and this can be regulated by a 2-in. globe valve on the pipe.

Sorting Table at Cobalt (By G. C. Bateman).—The sorting table illustrated in Fig. 67 was designed by Hugh Park, manager of the Nipissing mines, Cobalt, to handle a considerable tonnage at low cost and with no preliminary sizing or crushing, and at the same time separate the fines from the coarser material. In almost every mine in the district the fines are sufficiently rich to warrant shipment to a smeltery without concentration.

The table itself is a solidly built affair, 25 ft. long and 3 ft. 3 in. wide, and hung on a slope of 1:25 by iron rods from upright supports. The motive power is supplied by a simple little engine on the same principle as the cylinder and piston of a rock drill, and is known as the Park "pusher."

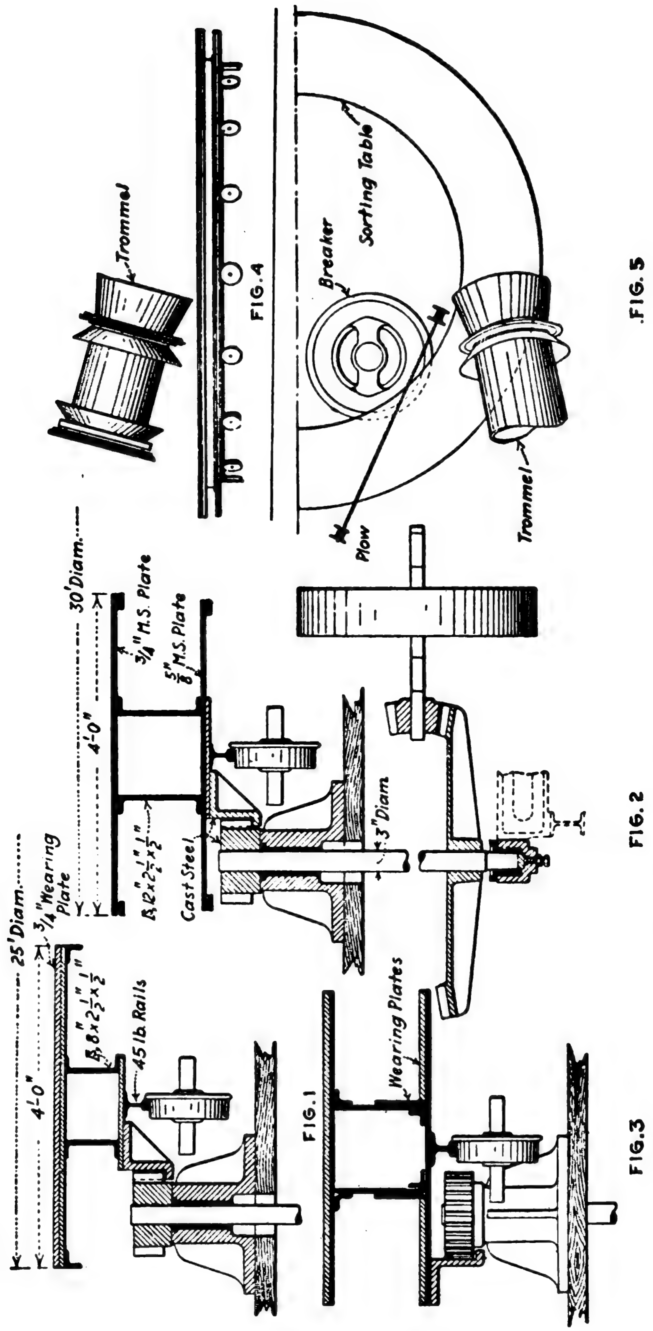


FIG. 68.—STRUCTURAL DETAILS OF RAND SORTING TABLES.

This is attached at the head end, and its only function is to pull the table up. At the end of the stroke, which is about 6 in., the air or steam is exhausted and the table falls back by its own weight and strikes against a solidly built bumper, near the upper end.

The rock as it comes from the mine is fed from a bin to the upper part of the table, which is covered for its entire length by a steel plate $\frac{1}{8}$ in. thick. Just below the feed there is a screen 4 ft. long, perforated with $\frac{3}{4}$-in. holes, which takes off all the fines. Beneath this screen is another, perforated with $\frac{1}{2}$-in. holes, which separates the smaller rock. These screens lead into separate bins. The oversize from the $\frac{3}{4}$-in. screen passes down the table till it comes to a depression where it is thoroughly washed by a spray. The spray consists of an iron pipe with two rows of small holes, drilled at an angle with the line of the pipe, the angle being opposite for the two rows. This gives a cross spray which thoroughly washes the rock. The water and slimes are carried off, through a screen, to a tank where the slimes are allowed to settle.

As the rock passes down the table from this point, the high-grade and second-class ore are sorted out by hand, while the waste is discharged over the end into a bin. The table bumps about 60 times per min., and when worked to its capacity can handle about 75 tons per day. Usually it takes from five to seven men to do the picking, depending on the richness of the ore. These tables have been in operation at the various shafts of the Nipissing for some time, and are also being in several other mines.

Types of Rand Sorting Tables.—Sorting is not extensively resorted to in the metallurgical works of this country as compared to those on the Rand, yet there are many mines where conditions are pointing to the advisability of rejecting as much waste as possible before the product of the mines is sent to the crushing machines in the mills. For this reason some of the sorting-house equipment which has been highly developed on the Rand is of interest not only to gold-mine operators, but to those operating lead and zinc mines, the coarse product from which it often pays well to sort by hand.

Sorting tables, while perhaps not so flexible as sorting belts, show a considerable number of good points which justify their use at present, states C. O. Schmitt in "A Textbook of Rand Metallurgical Practice, Vol. II." The use of a sorting table permits building a compact plant with only one broken-ore bin and of efficient supervision. There is no limit to the size of a sorting table beyond structural reasons, but the tables found in use on the Rand vary in diameter from 18 ft. to 30 ft., with a width of rim from 30 to 42 in. The width of the rim is limited by the reach of the sorting operators, and for that reason 42 in. should be the outside limit. The rim of the table usually moves at a peripheral speed

not exceeding 30 ft. per min., so that the men have ample time to see the waste present.

The diameter of the table is determined by the number of operators required to remove the waste rock; one man can pick out from 2 to 3 tons of waste rock per shift of 10 hr. Each man requires from 2½ to 3½ ft., so that the total circumference required can be calculated. In doing this, however, care must be taken also to provide for the space lost between the plow and point where the ore reaches the table plus any space required for washing, should this be done on the table. The space so lost varies from 8 to 12 ft., according to the arrangements made, and this must be added to the space occupied by the operators. The ore should be led to the table by a gate capable of making the distribution evenly over the whole width of the table, so as to facilitate both the washing and the subsequent picking out of the waste rock.

The waste removed from the table is generally thrown through holes in convenient places into a storage bin for removal to the waste dump.

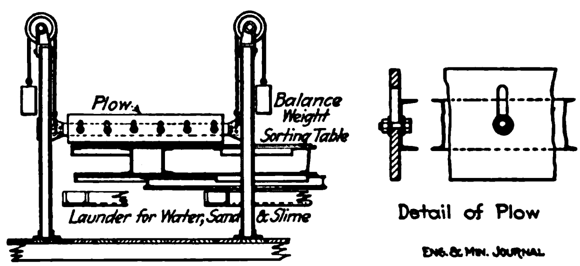


FIG. 69.—DETAILS OF SORTING-TABLE PLOW.

With a large table this becomes awkward, and to overcome the difficulty, sorting tables are now built with two decks, the lower deck serving to carry the waste to the opening of the waste-rock bin, where it is scraped off by a plow. The gold-bearing ore is left on the table until it has almost completed a full circuit, and it is then removed by a plow so as to make room for the fresh supply. Referring to Fig. 68, Fig. 1 shows the rim of a single-deck table, and Fig. 2 shows a table of the double-deck type. In Fig. 3 is shown a cross-section of the rim of an improved type of table having ample provision for wear in the shape of renewable plates.

A bin door with feed lip can be used for feeding the ore to the table, and is preferable to feeding directly from a washing trommel, as in that case there is a tendency to deliver the material in a narrow stream of greater depth than is desirable for good sorting. Whichever method is used, care must be taken that the direction of feeding is tangential to the rim of the table. Figs. 4 and 5 show the arrangement of washing trommel, plow, and fine breaker in relation to the sorting tables. The

angle between the plow and the table should be sufficiently acute to permit scraping off the ore with a minimum of power and the least possible wear on the face of the table. The plow must be rigidly held against the pressure arising from the accumulated ore, but it should be free to move vertically, for the reason that it should be possible to lift the plow whenever a piece of ore gets between it and the face of the table. To do this the plow is carried in suspension, and is well balanced. The wear on the plow must be taken into account by providing a wearing plate as shown in Fig. 69.

Sorting tables are generally carried on rollers by means of an ordinary steel rail fixed to the under side, and wherever a rigid foundation for these rollers cannot be obtained they should be carried in spring-borne bearings. Sorting tables of the type illustrated are driven by a circular rack and pinion from a countershaft arranged vertically, which is again driven by a set of bevel wheels from a horizontal shaft provided with a pulley. Direct driving by an electric motor is generally preferred, as the presence of line shafting and driving belts is not desirable in a sorting plant, being difficult to install, and interfering with the work. The power required varies with the size of the table, but in no case should exceed 5 hp. The sorting capacity depends upon the space available for operators but the carrying capacity is a function of peripheral speed and width of rim. With a velocity of 30 ft. per min. the carrying capacity of sorting tables can be taken as follows: The 30-in. table will carry 60 tons of ore per hr.; the 36-in. table, 72 tons; the 42-in., 84; the 48-in., 96 tons per hr. The cost of operating sorting tables varies considerably in the different plants, but may be taken as follows: Maintenance per ton milled 0.80c., power 0.16c., capital 0.20c., or 1.16c. total cost.

Sorting Belts Used in Rand Breaking Plants.—Sorting belts have been in use on the Rand for several years, and in view of their general utility they are now preferred to sorting tables. A sorting belt, states C. O. Schmitt in "A Textbook of Rand Metallurgical Practice, Vol. II," has the advantage over the sorting table in that its work is not limited to sorting only; and it can also be used for elevating the ore within certain limits. This is of considerable importance in plants where, either through lack of height in the headframe or through the use of preliminary breaking and washing trommels, the ore is delivered to the sorting appliance at a height not sufficient for feeding the fine breaker. The angle of rise of a sorting belt is limited to 10 deg., as otherwise the picking of waste becomes difficult, with a corresponding decrease in the efficiency of the labor. Apart from this, the height to which the ore can be elevated by the sorting belt is determined by its length, and this is again given by its tensile strength.

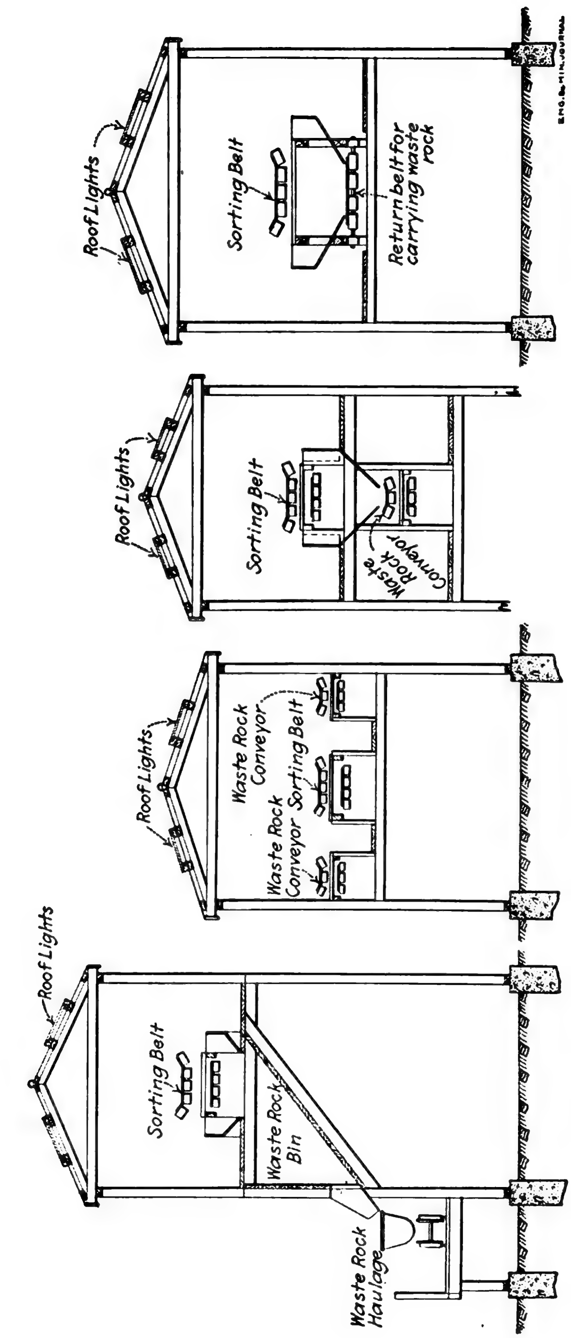


FIG. 1. FIG. 2. FIG. 3. FIG. 4.

FIG. 70.—FOUR METHODS OF DISPOSING OF WASTE FROM SORTING BELTS

The sorting belts in use vary from 30 to 48 in. wide, but for good work 42 in. should be the limit. The sorting belt is really a much enlarged conveyor belt using a special type of troughing idler. The belt may be made of rubber, reinforced by canvas, which is the type generally in use on the Rand. Canvas belts, however, are also in use, but in view of the large amount of moisture present preference is given to the rubber belt. The waste picked from the sorting belt can be dealt with in various ways as illustrated in Fig. 70. It can be dropped into a bin underneath the sorting floor, as shown in Fig. 1, for removal to the dump by trucks or other means. Another method in use is to place it on conveyor belts running alongside and on a level with the sorting belt, as shown in Fig. 2. However, with one waste belt only one side of the sorting belt is available for sorting, and in order to use fully the sorting belt two waste belts must be provided, as indicated in Fig. 2. If two sorting belts are used, only three waste belts are required, thus reducing capital outlay proportionately. A further method is shown in Fig. 3. In this case the waste belt is placed under the sorting belt and the waste need only be dropped through a hole in the floor; the operator therefore need not turn around, thus saving time and reducing the effort required. Another method of dealing with the waste rock is shown by Fig. 4. In this case the returning portion of the sorting belt is used as a carrier for rock.

The power required for driving sorting belts depends upon the amount of ore handled, the height to which it is elevated, and the length of the belt; it should not exceed 10 hp. in any case.

The sorting capacity depends upon the length of the belt, that is, upon the number of picking hands for whom there is room, from 2½ to 3½ ft. being the allowance per man on one side. The carrying capacity, if not overloaded, is somewhat less than that of a sorting table of the same width, and can be taken as follows: A 30-in. sorting belt will carry 54 tons per hr.; a 36-in. belt, 66 tons; a 42-in. belt, 78 tons, and a 48-in. belt, 90 tons per hr. Temporarily these capacities can be largely exceeded at the expense of efficient sorting, as the ore will then be packed in too deep a layer to permit of efficient work.

The length of the sorting belt is limited by the tensile strength of the material used for the belt, as has already been stated, and also by the number of operators needed to remove the waste. Assuming a case where 1200 tons of ore and waste are supplied from the shift of 10 hr., of which one-sixth or 200 tons, is waste; and further, that 33%, or 400 tons, will disappear when passing over the screening arrangements, so that 800 tons will pass over the sorting belt, equal to 80 tons per hour, a 42-in. sorting belt will be required. To remove the 200 tons of waste, from 67 to 80 sorters will be required, for whom up to 240 ft. of sorting space must be provided, or 120 ft. on each side of the belt. Adding to this

8

15 ft. for washing purposes, the total length of the belt becomes 135 ft., which is well within the limit enforced by the tensile strength of the belt.

The life of a sorting belt is usually not long, depending on its length, the quality of the material used, and the method of feeding the ore to the belt. If the ore is not fed on in a broad even stream, as, for instance, when feeding from a washing trommel, the belt is liable to be loaded more on one side than the other, resulting in uneven running and rapid wearing out of the edges against the side idlers. The cost of operating sorting belts may be taken as follows: Maintenance per ton milled, 0.40c.; power, 0.16c.; capital charges, 0.24c.; or 0.8c. total cost.

Intermittent Sorting Belts.—At the No. 5 shaft of the Crown Mines, Ltd., on the Witwatersrand, the ore is delivered from the skips to two sets of grizzlies, the upper of which is spaced wider than the lower. The oversize from both grizzlies, 2 in. and up, is delivered into bins whence it is fed to the sorting belts. It is washed, as it is fed, in a chute fitted with sprays and having a bottom fitted with grizzlies of fire-bar section to remove the coarse particles, slime and water. The wash water is delivered by a high-lift, 5-in. centrifugal pump supplied from the mine-pump column.

The four sorting belts are 36 in. wide, have an inclination of 14 deg. and are run at a speed of 150 ft. per min. Each belt is driven by a 25-hp. motor running at 500 r.p.m. through a 60 to 1 reduction gear to the head pulley of the belt. By means of a friction clutch, the belts are stopped and started at will. The object of this is to provide for a thorough sorting of the belt contents in sections. A belt full of rock is run out and stopped. It is then the duty of each sorter to pick the belt clean of waste immediately in front of him. When the overseer is sure the belt is clean, he marks the point up to which it has been picked, sets the belt in motion, and stops it again when the mark reaches the limit of picking on the other end. The waste is placed on the under side of the belt and delivered into a bin. One sorter is employed on the waste side of each belt to sort back any ore that may have been accidentally discarded. The tube-mill pebbles are also picked out and deposited on another belt situated under the sorting belt and running diagonally across and under the sorting floor so as to be accessible from all four sorting belts.

Concentrating High-grade Fines by Hand (By A. L. Flagg).—In the early history of a mine producing high-grade ore, all or a part of which is of shipping quality, it is frequently a problem to know just what disposition to make of the fine material. This is especially true if it is attempted to keep the mine self-supporting until a plant can be installed to treat all the ores, depending in the meantime on the revenue derived from shipments for all or a part of the development fund, or even setting aside

such income as a sinking fund for the purchase of the necessary mill equipment.

A mine which can pay for its own development and in addition build a mill for itself is indeed rare. However, such mines do exist. It is the purpose of this paper to describe a method adopted to make a marketable product out of the fine material taken from a high-grade silver mine in Mexico during the early stages of its development. The mine was nearly a hundred miles from a railway. In addition to the charge of $20 per ton for muleback transportation, there was an additional railway freight charge of nearly $5 per ton to the smelter. To this must be added the smelter's charges, assaying and commissions for liquidating at the smeltery, taxes and the cost of mining. To stand all these charges and still pay a profit the ore had to be high-grade indeed. As broken in the mine the ore under consideration ran from 40 to more than 1000 oz. of silver per ton, with a small amount of gold. Almost invariably the higher the silver content, the greater the percentage of fines produced in the mines.

At the period when this process was put into use the mine was in the prospect stage. A small, single-compartment incline shaft was being sunk on the vein with drifts each way at intervals of 50 ft. As a rule the ore was exceedingly hard, but it carried many minerals rich in silver and extremely brittle. The predominant minerals were galena, proustite, pyrargyrite and stephanite, besides native silver in wires and flakes. The gangue was quartz and rhodonite with local occurrences of calcite. Zinc and iron sulphides also occurred as accessory minerals. In instances where quartz or calcite predominated as the gangue mineral, every round fired would produce a large amount of fines composed of the more brittle components of the ore. The result was that frequently 20% of the mine-run would pass a 1-in. screen and assay from $60 to $150 per ton.

As each carload of ore was brought from the mine, it was dumped on the patio. There it was sorted by boys into: (1) Mill ore, which went to a stack, to await the erection of a mill; (2) shipping ore, which went through the usual sorting process common in such instances; (3) fines, that is, material which would pass a 2-in. ring. In some cases where the face showed exceptionally brittle ore which was correspondingly high-grade, the fines were put in sacks to lessen the risk of loss from rehandling.

After all the coarse material had been picked out, the fines were shoveled over a screen having a 1-in. opening. Of the oversize, about 4% was picked out as shipping ore, the rest being sent to the mill stack. The undersize was all treated in a two-compartment hand jig. While some material of a grade suitable for shipping was obtained as coarse concentrates from the two compartments, the real value of the jig lay in the classification between the coarse material, which remained on the screens and the high-grade hutch product.

The overflow water from the jig was run through settling tanks to save the slimes. Just what disposition can be made of these slimes is at present a problem. It is quite probable that they can be mixed successfully with the mill feed in small quantities and not cause any trouble. The amount of slimes saved was small, being about 1% of the whole, but they were worth $50 per ton.

The coarse concentrates were removed by skimming; the tailings were dumped from the boxes after litting them out of the jig. While the amount of ore won in this way was comparatively small, it was sufficient to justify the expense. In the period of test, the average amount obtained this way was 81.4 lb. per day per man, running 534.58 oz. of silver per ton.

The frequency with which the hutch product was removed for retreatment on a *planilla* varied with the percentage of fines produced. As a rule, it was necessary to clean out the hutch every third day, sometimes oftener. To do this, the screen boxes and plunger were removed and the material taken out with a short-handled shovel.

The *planilla* requires some skill to operate. In this instance it consisted of a shallow box 10 in. deep, 4 ft. wide, 6 ft. long, open at one end, and resting on the ground, with a slope of not over 3 in. in its length. Directly in front of the lower open end, a soap box was sunk even with the ground surface. A 1-in. cleat, 6 in. from the end and extending to within 1 in. of the sides of the box nailed across the bottom completed the device. A small pipe leading from the creek kept the soap box full of water.

After piling about 25 lb. of the hutch product against the back of the *planilla*, the operator takes his position over the soap box with a small, oval, wooden bowl, called a *batea*, in his hand. With this he scoops up water from the soap box and throws it on the *planilla* heads all in one motion. After a while it is necessary to scrape the concentrates back into the end of the box as they wash down with the sands. The sands are carried off by the wash water past the ends of the cleat at the open end of the *planilla*. The washing is continued until a remarkably clean prod-

TABLE XVIII.—RESULTS OF HAND CONCENTRATION

	Pounds	Oz. of Ag per ton
Undersize from 1-in. screen, jig heads......	39,304	172.9
Coarse jig concentrates, shipped...........	1,760	582.2
Coarse jig middlings and tailings, stacked..	17,732	58.2
Hutch product..........................	9,680	307.0
Planilla concentrates.....................	6,776	534.6
Planilla tailings.........................	2,904	146.4
Planilla slimes..........................	55	26.5
Jig slimes..............................	15	105.9

uct is obtained, this being dried and sacked for shipment as concentrates from any table would be.

A careful record of production and costs was kept, covering a period of about six months. It was found that with the labor of the jig men at \$0.50 a day, clean concentrates could be produced at a cost of \$6.16 per ton, exclusive of mining. The average amount of concentrates produced daily over the period of test was 136.4 lb., of which 81.4 lb. was a coarse jig product, while the remaining 55 lb. was from the *planilla*. Table XVIII expresses concisely the results obtained.

CLASSIFIERS

Pockets and Cones Using Hydraulic Water

A Simple Vortex Classifier.—A simple type of vortex classifier is used with success in some of the mills of southwestern Missouri. The

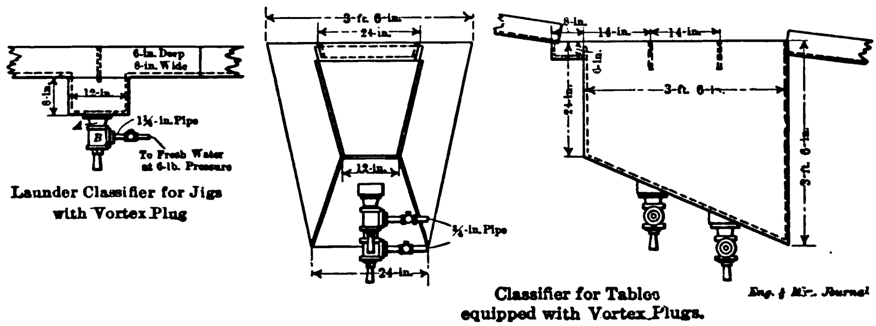


FIG. 71.—VORTEX PLUGS ATTACHED TO LAUNDER AND TABLE CLASSIFIERS.

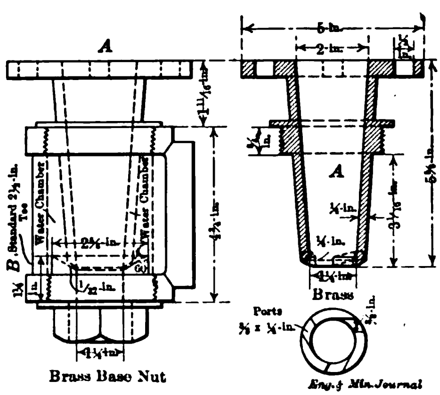


FIG. 72.—VORTEX PLUG IN JOPLIN CLASSIFIERS.

classifier can be used either in the pockets of a launder-type classifier or in a deep-pocket classifier as is shown in Fig. 71. In the pocket form the pulp is allowed to build up to give the proper slope to the

bottom, whether there be one or more hydraulic attachments in the pocket. In both types a baffle board is used above the pocket proper.

The details of the hydraulic attachment are shown enlarged in Fig. 72. The attachment consists of a brass casting *A*, which fits into *B*, a standard 2½-in. tee which is fitted with a bushing so as to take the sorting water from a ¾-in. pipe. The water from the water chamber formed by the tee passes through vortex openings in the bottom of the casting *A*, the brass fitting *C* being screwed into the bottom of the tee until there is only about

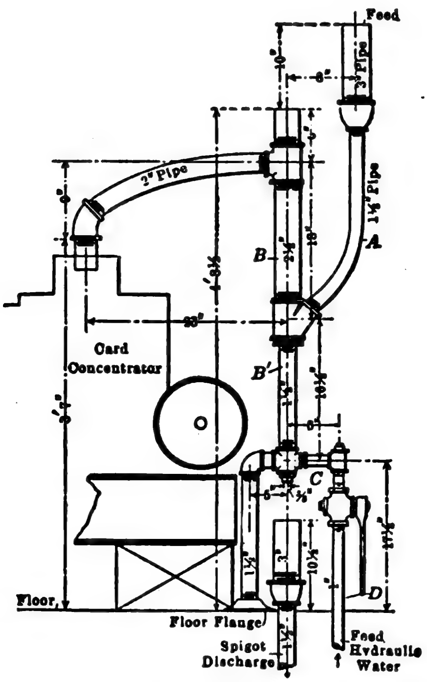


FIG. 73.—BUNKER HILL & SULLIVAN PIPE CLASSIFIER.

$\frac{1}{32}$-in. opening between the casting *A* and the nut *C*. The pulp that settles past the vortex current is drawn off through the fitting *C*, the flow being regulated by means of wooden plugs of which several are provided with different size holes bored through them. This whole attachment bolts to the bottom of the classifier proper. All parts of this classifier are cheap and easily replaced.

Moreover, the classifier is capable of a certain amount of adjustment in regard to the vortex by changing the distance between the interior parts. A peculiarity of the vortex is that it is directed downward. In this way clogging from shut-downs is less liable to happen, while the

ease with which the whole device can be taken apart permits its being cleared readily in case it should become clogged.

Pipe Classifier in Bunker Hill & Sullivan Mill (By John Tyssowski).—In the new concentrating mill of the Bunker Hill & Sullivan Mining and Concentrating Co. at Kellogg, Idaho, the product from the first hutch of the classifying jigs goes to pipe classifiers. The overflow from these pipe classifiers passes to Card concentrating tables, the dewatered spigot product being shipped directly. The pipe classifier is ideal for the conditions under which it is operated, *i.e.*, for handling a hutch product under 2 mm. in size and consisting mostly of clean galena ore. If the feed contained a middling product this classifier would not be satisfactory, as the middlings would be discharged with the spigot product. The pipe classi-

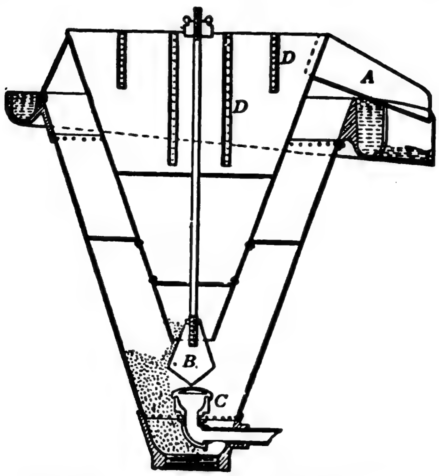


FIG. 74.—MALCHUS HYDRAULIC CLASSIFIER.

fier raises the percentage of lead from 50% in the feed to 80% in the spigot material. Feed water under 20 ft. head is used on the classifiers. As shown in Fig. 73, the classifying machines are extremely simple in construction, and can be readily built up from sections of 1-, 1½-, 2-, 2½- and 3-in. pipe, and suitable connections. Referring to the drawing, the feed is through *A*, *B–B′* is the sorting column, *C* the spigot and *D* the pipe through which the hydraulic water is supplied. The various portions of the classifier have the following volumes, expressed in gallons per min: *A*, 5.8; *B*, 9.2; *B′*, 3.3; *C*, 4.7; *D*, 8. The sorting velocity is about 7 in. per sec. The pipe classifier is peculiarly adapted to the treatment of the heavy silver-lead galena ores of the Cœur d'Alene district, and has given satisfactory results in the Bunker Hill & Sullivan mill.

The Malchus Hydraulic Classifier (By Warren C. Prosser).—The Malchus classifier is the invention of a San Juan mill man who is now in

charge of the demonstrating operations of milling the argentite ore of the Intersection mine. The classifier was designed primarily to treat galena and gray-copper ores which had been slimed as it was found that the ordinary classifiers were unsuccessful with these ores, the slimes being lost in the tails and middlings from the concentrators. In this classifier, referring to Fig. 74, the slimes pass out through a launder *A* instead of through the outer cone and can be treated separately. The adjustable double-cone plug *B*, which regulates the feed, insures all pulp being subjected to the upward current. By using spray cups *C* with different-size slots, and the cone-plug arrangement, a wide range of classifying action can be covered. The baffle board *D* is designed to cut out slimes only and to diminish the flow of water necessary for successful operation.

Boston Consolidated Classifier.—The Boston Consolidated mill at Garfield, Utah, was noted among millmen for its excellent classification. Classification was accomplished in a somewhat modified form of the well-

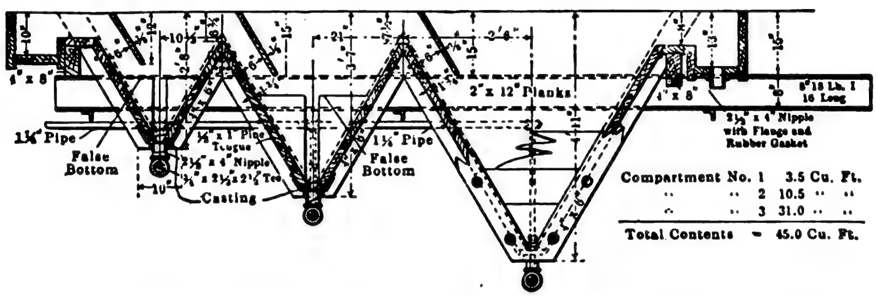


FIG. 75.—THREE-COMPARTMENT HYDRAULIC CLASSIFIER WITH FALSE BOTTOMS AT BOSTON CONSOLIDATED MILL.

known Anaconda, three-compartment classifier. Originally the first two compartments gave trouble through the backing up of pulp on the sides. After accumulating for a while, it would come down with a rush destroying the equilibrium of the conditions in the compartment. To avoid this a false bottom was put about half way up the first two compartments; the lower part, where the jet current was introduced, was continued up to this floor as a neck or box compartment about 4 in. square. The sands then built up on the false bottom and formed a floor to the compartment at the angle of repose of the material. As the false floor is wide enough so that the box compartment is outside the slope of repose, trouble from rushes of pulp in the compartments is eliminated. The pulp going to the last compartment is so fine that it gives no trouble. Fig. 75 shows the details of the classifier used at the Boston Consolidated mill. The false bottoms of the first two compartments are indicated

The character of the work accomplished by the classifier is shown by the screen analysis of the spigot products given in Table XIX. The feed

TABLE XIX.—SCREEN ANALYSIS OF SPIGOT PRODUCTS

Mesh	Relative weights of spigot products		
	First spigot	Second spigot	Third spigot
On 50	42	2	0
On 80	38	28	6
On 100	8	12	8
On 150	6	20	22
On 200	2	8	10
Through 200	4	30	54

to each classifier came from six Nissen stamps crushing through a diagonal-slot punched screen, equivalent to a 26-mesh, No. 26 wire screen, at a dilution of about 7.22 parts of water to one of dry ore. The ore crushed by the stamps varied greatly, and averaged 1 month slightly more than 9 tons per 24 hr. per stamp. At that rate of feed, the classifiers worked quite satisfactorily, so that a classifier with the dimensions given in the article referred to is able to handle at least 18.5 tons of pulp per hr. without crowding.

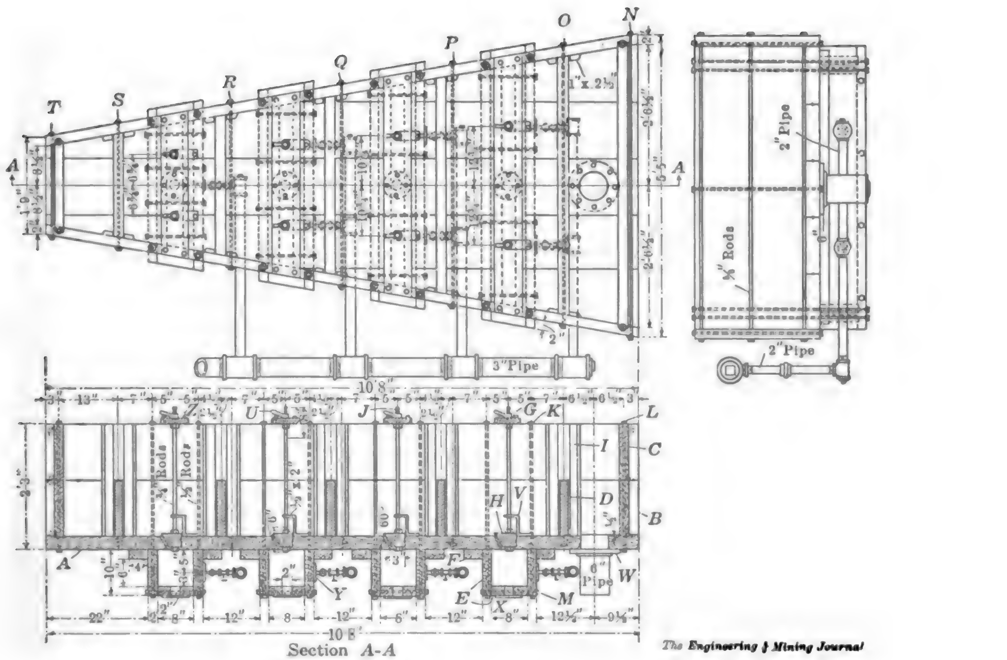


FIG. 76.—DETAILS OF THE YEATMAN CLASSIFIER IN NEVADA CONSOLIDATED MILL.

The Yeatman Classifier.—The classifier shown in Fig. 76 was devised by Pope Yeatman several years ago for use in the Missouri lead district,

and is mentioned in Richards' "Ore Dressing." It is used at the Steptoe concentrator at Ely, Nev., where, in order to increase the capacity, some minor changes have been made, such as providing two rods for moving the plug of the slot admitting the classifying current, and modifications in the manner of withdrawing the concentrates. As the drawings in Richards' book are lacking in details, it has been thought of interest to publish the accompanying drawings, which show the Yeatman classifier in its latest form.

TABLE XX.—BILL OF MATERIAL FOR YEATMAN CLASSIFIER

Number required	Mark	Article	
5	A........	3 × 12 in. × 11 ft............	Bottom.
4	B........	2 × 14 in. × 11 ft............	Sides.
2	C........	2 × 14 in. × 7 ft............	Ends.
1	D........	1½ × 12 in. × 17 ft............	Baffles.
4	E........	2 × 10 in. × 18 ft............	Box.
2	F........	2 × 4 in. × 18 ft............	Nailing strip.
1	G........	1½ × 8 in. × 16 ft............	....
1	H........	3 × 5 in. × 14 ft............	Bevel gate.
1	I........	1 × 6 in. × 11 ft............	Cleats.
8	J........	¾ × 32-in. rods...............	Bevel gate.
16	K........	½ × 39-in. rods...............	Box.
5	L........	½ × 29-in. rods...............	Ends.
16	M........	½ × 14-in. rods...............	Bottom.
3	N........	⅝ × 68-in. rods...............	End.
1	O........	⅝ × 63-in. rods...............	Bottom.
1	P........	⅝ × 54-in. rods...............	Bottom.
1	Q........	⅝ × 46-in. rods...............	Bottom.
1	R........	⅝ × 37-in. rods...............	Bottom.
1	S........	⅝ × 29-in. rods...............	Bottom.
3	T........	⅝ × 24-in. rods...............	End.
8	U........	Lock nuts.....................	Bevel gate rod.
8	V........	Guides........................	Bevel gate rod.
1	W........	6-in. pipe flange.............	....
4	X........	2-in. pipe flange.............	....
7	Y........	1-in. pipe flange.............	....
16	Z........	⅝-in. lag screw...............	....
90	....	½-in. hex. nuts...............	....
74	....	½-in. M. I. washers...........	....
16	....	½-in. cut washers.............	....
24	....	¾-in. hex. nuts...............	....
22	....	⅝-in. M. I. washers...........	....
32	....	¾-in. cut washers.............	....
8	....	¾ × 5-in. bolts...............	....
16	....	⅝ × 3½-in. bolts..............	....
28	....	7/16 × 3-in. bolts............	....

All rods marked J. to be threaded 6 in. on one end and 8 in. on other end.
All other rods to be threaded 2 in. on each end.

The main feature of the classifier is that the classifying water is admitted in a sheet through a slot rectangular in cross-section, of variable width, and extending clear across the bottom of the classifier. This results in an even upward current throughout the whole cross-section and allows the pulp, as it settles, to spread out in a broad band, giving better opportunity than in the ordinary cone classifier for the undersize particles to be washed free from those of the right size and density to fall through the classifying current. The classifier is allowed to build up its own bottom; all joints are plain and square, so that there is no trouble in keeping the box water-tight. This classifier is comparatively cheap to make. A piece of hardwood is used for the plug regulating the width of the opening for the entrance of the classifying water. Table XX gives the bill of material for the Yeatman classifier.

At the Steptoe mill one of these classifiers is provided with a glass side, so that the manner in which the pulp slides down the built-up bottom of settled ore particles to the classifying opening is well shown, and also the region of turmoil and agitation, just above the slot where the particles are being classified in the upward current of water coming from the water-box below.

The advantage of this form of slot classifier is that it does its work with a minimum of attention. The fault of most classifiers is that they require attendance in order that they may do good work and the cause of the poor classification at most mills is the little attention given to the classifiers. The attention of the few men that are required in a large mill is pretty well taken up by the other machinery, and ordinarily it is desirable to have a classifier that will do fairly accurate work with little watching rather than an exceedingly accurate machine that must almost constantly be readjusted. Lately the Nevada Consolidated company has been experimenting with the Janney classifier. This device, when given close attention by a man who understands its operation, works admirably and does better work than the Yeatman classifier, but unless closely watched it gets out of adjustment more quickly and then does poor work.

Cones and Tanks not Using Hydraulic Water

The Diaphragm Cone Classifier.—The Caldecott diaphragm cone classifier illustrated in Fig. 77, was introduced into South African ore-treatment practice in 1908. It differs from other types of hydraulic classifiers in that it must be operated while filled or partly filled with settled solids, instead of with the usual fluid pulp carrying ore particles in suspension and rapid movement. The diaphragm is a support for the settled solids, and is placed near the bottom of the cone at a little distance from the sides so that the solids may gradually pass down the annular

space around the periphery of the diaphragm to the outlet at the apex of the cone. As a rule the diaphragm consists of a plain disk of wood or thin sheet steel, conveniently supported as shown in the illustrations.

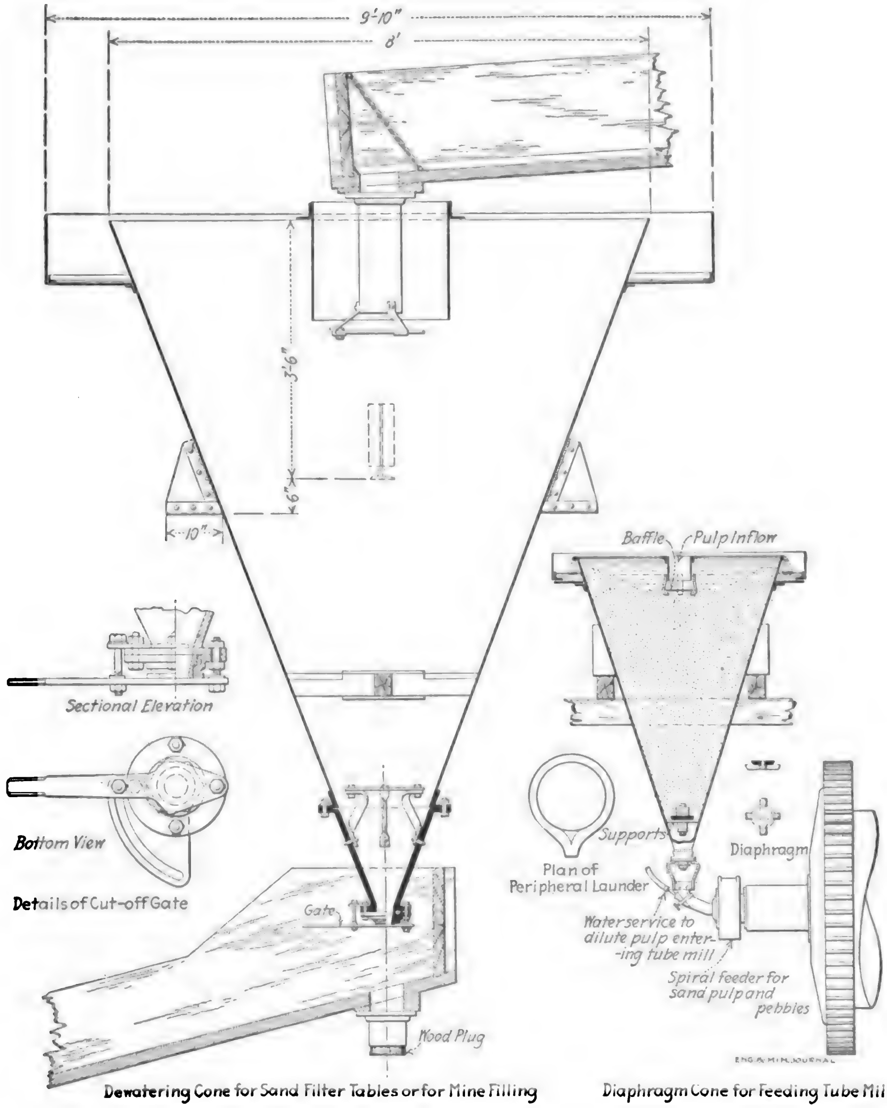


FIG. 77.—APPLICATIONS OF THE CALDECOTT DIAPHRAGM CONE.

Conical, perforated, and curved forms have been proposed, but in practice these show no special advantage. The simplicity and cheapness of the device renders an adjustable form hardly necessary, since a few trials

with wooden disks will speedily result in the most suitable diameter and position being ascertained for any existing set of conditions.

In general it may be said that the coarser and more granular the pulp fed to a diaphragm cone, the greater its capacity, but with a fine pulp containing much colloidal slime its capacity is less, and owing to the tendency of the settled solids to cohere and bridge over the annular space, this space must be made wider by placing the diaphragm higher in the cone. Unless this is done, the function of the diaphragm in preventing a single vertical channel from forming in the mass of settled solids down which the pulp descends, cannot be performed and its efficiency is in consequence impaired.

The diaphragm cone is largely used in South Africa for separating coarse sand from the pulp for grinding in tube mills. The standard cone for this purpose on the Rand, for a 5½ × 22-ft. tube mill is 6 ft. in diameter by 9 ft. deep with a flat disk diaphragm 8 in. in diameter, and so placed in the cone as to have an annular space around it 2½ in. wide or more, according to the more or less slimy nature of the crushed ore.

The circular outlet at the apex of the cone, in the shape of a replaceable cast-iron nozzle, is 1½ to 2¼ in. in diameter, and yields an underflow of 200 tons to 400 tons of solids per 24 hr., as a pulp containing about 27% moisture. A cut-off gate, shown in Fig. 77, is used to regulate the underflow to any further degree required, or to stop it entirely. Owing to its viscous nature, the underflow must be allowed to fall freely from the cone, as any attempt to change its direction by means of a curved pipe from the outlet merely results in choking. This thick underflow issues at a slow velocity, and when large tonnages are fed to the tube mill, is usually diluted with water or solution, before entering the tube mill, to a pulp containing about 40% moisture. It is due to the slow velocity of the underflow that so large an outlet can be used, and this practically obviates any risk of choking by chips of ore or accidental refuse to which ordinary classifiers with fluid underflow and small outlets are liable.

Smaller cones can of course be used, but a sufficient superficial settling area is always desirable to allow all particles requiring regrinding to settle, as otherwise return-sand, safety cones for the overflow pulp are necessary. In most cases on the Rand the tailing pulp overflowing from the classifiers must be delivered at a considerable height for subsequent cyaniding, so that there is usually ample fall available for the installation of cones of any desired dimension. The use of a large cone is not detrimental from the classification standpoint, as the settled solids form a basin of greater or less diameter, according to the volume of inflowing pulp, which serves as an automatic regulator and insures a uniform and constant underflow. If a coarser pulp than usual is delivered to the classifier, the coarseness of the underflow and overflow is correspondingly in-

creased, and the tonnage of the former is also increased, while its moisture is slightly decreased. A diaphragm cone, run so full of settled solids that the out-flowing pulp stream near the periphery is hardly ¼ in. deep, is practically a center-discharging buddle, with the consequent efficacy of that well-known metallurgical appliance for retaining pyrite for further comminution.

The grading analysis, Table XXI, illustrates the nature of the classification effected by the cones for separating sand for return to tube mills, the underflow of the conês entering the tube mills and the overflow passing to the cyanide plant.

TABLE XXI.—GRADING ANALYSIS OF CONE PRODUCTS

	+60	−60 +90	−90
Underflow	67.1%	20.7%	12.2%
Overflow	7.4%	15.9%	76.7%

Large cones, 8 ft. diameter by 10 ft. deep, are used on the Rand to separate the leachable from the unleachable sand in the pulp, the sand going to the Caldecott sand filter tables for further dewatering the thick sandy underflow. The diaphragm cone was indeed originally devised as an accessory to these filters to insure their receiving a large steady volume of sandy pulp containing a minimum of slime and moisture. Table XXII shows the grading analysis of the pulp which was classified into about equal tonnages of sand and slime while treating a very large amount of crushed ore per day.

TABLE XXII.—GRADING ANALYSIS OF CLASSIFIED SAND AND SLIME

Screen	Sand, per cent.	Slime, per cent.
+ 60	8.4	......
+ 90	33.3	......
+ 200	46.3	2.21
− 200	12.0	97.79
	100.0	100.00

While the operation of these large cones is the same in principle as of the tube-mill cones, the fine-slime overflow renders larger cones necessary to settle all the sand, and the underflow of sand pulp contains about 30% moisture. Two diaphragms will be observed in the figure, the second one being used, when much colloidal slime is present, to still further check the tendency of the settled sand to pack and bridge over the discharge orifice. The efficiency of the diaphragm for affording an underflow almost free from slime is due to the small amount of moisture present, to which suspended slime is proportional, since with a tailing pulp of seven of water to one of solids and containing 60% sand, 96% of the total water

passes away with the slime in the overflow, while the underflow contains 30% moisture or little more than the water in the interstitial spaces of the settled sand. By varying the number of these cones and by diluting or not the underflow with water before entering similar cones so as to wash out more very fine particles, the percentages of sand and slime separated can be varied within wide limits.

These large cones are also extensively used for dewatering cyanide-plant tailing before pumping or gravitating through a borehole into a mine for filling purposes. The surplus water in the pulp enters the diaphragm cones, overflows and is returned in circuit to the pump to serve for bringing up more sand. This system has been in satisfactory use during the last year at the Simmer & Jack mine.

Improved Caldecott Cone.—W. A. Caldecott, of Johannesburg, has patented (U. S. No. 1,008,524) the use of a circumferentially notched plate to assist in the settlement of solids in overflow classifiers. The object is to avoid the holes or passages which form in the ordinary classifier in the solids, which result in sudden rushes of the liquid through the aperture in the bottom of the vessel. In large tanks, there may be perforations in the plate, as well as circumferential notches.

The Michel Hydraulic Classifier.—A new type of classifier has recently been introduced in Central America and Brazil, with, it is believed, satisfactory results. It is the invention of G. Michel, of Paris, and constitutes a distinct departure from classifiers involving the use of cones, spitzkasten, spitzlutten, etc. Before describing the apparatus, it may be interesting to refer briefly to the principle on which it is based. In the case of sand containing several constituents of different densities, it is the custom in mechanical preparation, to class it according to size by the aid of screens. Examining the law of the free fall of bodies in water, it is found that all bodies falling freely in a medium of density X acquire a velocity v, which at the beginning of the fall is represented by the formula

$$\frac{dv}{dt} = \frac{g(1 - X)}{D}$$

D being the density of the grain. When the speed becomes uniform if the height of the liquid column permits this, this uniform speed becomes in water

$$v = 2.44a\ (D - 1),$$

a being the diameter of the hole through which the grains will pass. Table XXIII shows the speeds in meters per second of certain grains at the beginning of the fall and after certain periods.

TABLE XXIII.—SPEED OF MINERAL GRAINS FALLING IN WATER

Diameter in mm.	Nature of grains	⅛ Sec.	¼ Sec.	½ Sec.	1 Sec.	2 Sec.
	Galena.........	0.903	1.441	1.630	1.650	1.650
15	Pyrites.........	0.825	1.174	1.287	1.293	1.293
	Quartz.........	0.570	0.767	0.801	0.817	0.817
	Galena.........	0.704	0.814	0.823	0.824	0.824
4	Pyrites.........	0.586	0.643	0.646	0.646	0.646
	Quartz.........	0.383	0.409	0.409	0.409	0.409
	Galena.........	0.409	0.413	0.414	0.414	0.414
1	Pyrites.........	0.321	0.323	0.323	0.323	0.323
	Quartz.........	0.203	0.204	0.204	0.204	0.204

It will be seen that it is necessary, in order to obtain a constant speed of grains of the same size, to have a liquid column of constant section and of a sufficient height. If in carrying out the inverse operation one gives to the liquid column a uniform vertical movement directed from the bottom to the top, and a convenient height to enable the grains of largest size to obtain their constant speed, an apparatus is obtained by which it is possible to carry over those portions of light density and to collect at the bottom of the liquid column the grains of largest specific gravity.

In practice it is not sufficient, in order to separate the larger grains by difference in density, to utilize simply a liquid column in movement, because of the necessity of reducing to a minimum the eddies, and also the friction against the boundaries of the column. It is necessary to employ a cylindrical column of appropriate diameter to avoid sharp bends and present the least surface of resistance, and also to insure a sufficient section to minimize the friction of the grains against themselves. In hydraulic classifiers dealing with grains of comparatively large diameters it is hardly necessary to take much notice of the phenomenon of surface tension, but this point becomes important when dealing with fine sand. Its effect is due to the fact that the surface of a liquid in contact with air or any other gas presents a resistance to rupture and grains having a density higher than that of the liquid actually float on the surface without being able to break it. To avoid the inconvenience of the surface-tension effect it is necessary to introduce the grains which are to be classified into the interior of the liquid column so that they may break the pellicule forming its surface. Moreover, to maintain a constant speed of the liquid column and to avoid the production of eddies it is indispensable to accomplish the removal of the classified products in such a way that there are no eddies set up and no alteration in the speed of the column. It is also indispensable for the same reasons to maintain a constant feed of the material classified and of the liquid.

These considerations being noted, reference may be made to Fig. 78, showing the construction of the Michel hydraulic classifier. It will be seen that it consists of a vertical cylinder *C* of appropriate height and diameter, at the center of which is fixed a tube *T* in communication with a hopper *H* in which the mixture of water and mineral to be classified is placed. Between the tube and hopper is placed a regulating valve and the liquid containing the mineral grains in suspension passes first down the tube *T* onto a cone *L* and then upward again through the outer portion of the cylinder *C*. The grains are cleared of mud and fine sediment, and the latter particles are carried up with the constantly upward flowing water to an annular trough C_1 from which a regulated

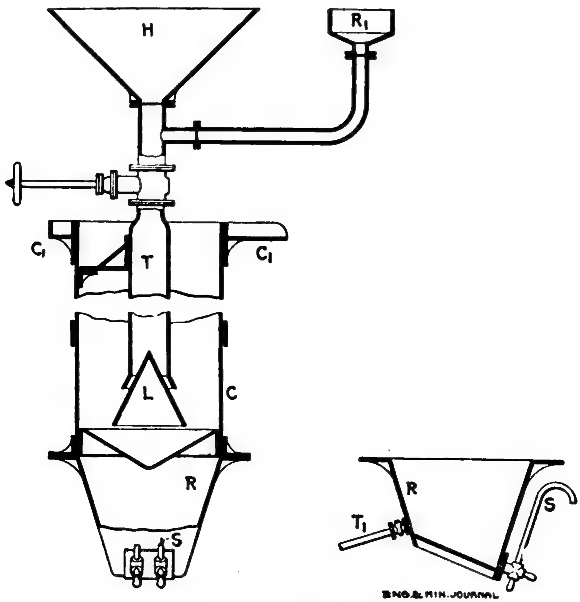


FIG. 78.—THE MICHEL CLASSIFIER.

discharge is effected. The particles of heavier specific gravity separate and fall on an inverted cone at the base of the cylinder *C*. This cone has numerous holes in it whereby the classified particles fall through into a reservoir *R*. This reservoir is furnished with one or more siphons *S*, by means of which the classified matter is quietly and regularly withdrawn. In order to compensate for the water which is drawn away in this manner along with the classified particles, a pipe T_1 is attached to the receiver by means of which a stream of compensating clear water is introduced simultaneously with the withdrawal operations of the siphons. In this way disturbance of the main flow is avoided, and there are no eddy currents or speed alterations. The cylindrical form of the

apparatus allows material in movement to obtain and preserve constant speed of rising for each grain during the whole of its upward course. The cone L has the function of uniformly spreading the flow throughout the whole circumference of the cylinder and in conjunction with the valve has also an effect in regulating the speed. In order to further insure a constant rate of feeding, an auxiliary reservoir fitted with a float valve R_1 is attached to the central tube. With this provision it is found that the Michel classifier is capable of doing good and uniform work, and is said to have been adopted with success by the Darien Gold Mining Co., of Central America, and by the Conquista Xicao Gold Mines in Brazil.

Piping for Callow Cone Installations.—As with other devices used to settle or thicken pulp, the discharge pipe on the Callow cone is liable to become stopped up, although this tendency is lessened by bringing the discharge opening as near level with the height of the feed as is practical, thus admitting of the use of a large discharge opening. This clogging is especially apt to occur when the feed to one of the devices fed by the Callow cones has to be shut off. At the Ohio Copper Company's mill, William Kidney, superintendent, has devised an ingenious way of arranging the piping of the Callow cones so as to facilitate their starting after a shut-down.

The discharge pipes from the Callow tanks discharge into funnels on the feed pipes of Wilfley tables, at a height only a little below the solution level in the tanks. The funnel breaks the continuity of the discharge pipe and ends the siphoning action. Each of the Wilfley tables takes the feed from two Callow tanks. There are two cross pipes at each table. Clear-water pipes are carried along beside the Callow tanks just a little higher than the level of the discharge pipes from the tanks. These pipes provide the water for washing the floors and for starting the cones when they get stopped up. A T-connection is made between the lines where a discharge pipe crosses the line of the water pipe and a valve is put on the connecting pipe. When a discharge pipe becomes stopped up, the valve connecting with that discharge pipe is opened slightly and by placing the hand over the discharge opening of the Callow pipe the water is forced up through the discharge pipe of the Callow cone into the tank itself, and the pulp is broken loose. Thus the tank is made ready for work without any dirtying of the floor. It is an easy and effective way of doing the work, and is especially useful where a number of Callow tanks are employed.

Joplin Intermittent Settling Tank (By Claude T. Rice).—The flat topography of the Joplin district makes elevating necessary if cone continuous settlers are used for the table feed. It is more common practice, therefore, to use a flat settling box which works intermittently. The construction of these is characterized by a good deal of ingenuity.

They are usually built in pairs with their bottoms sloping diagonally to the discharge from the opposite corner. Fig. 79 gives the essential features of the tank. Proper dimensions would be about 15 × 20 ft. for each half, as shown, with the bottom sloping about 3 in 20. This determines the length, since only a certain amount of fall can be allotted to the box in the layout of the plant. The width is in turn determined by the length, as the compartment should be about square to get the best sluicing effect.

The tank sides and bottom are built double of 1-in. plank and have strips of ball wicking inserted which catch the silt and thus form a tight box. The frame of the tank is made of 2 × 6-in. joists and studding.

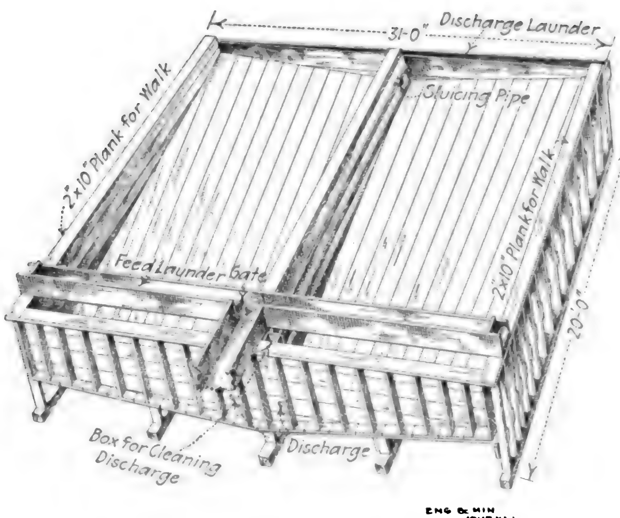


FIG. 79.—TWO-COMPARTMENT SETTLING TANK FOR PROVIDING UNIFORM TABLE FEED.

The central partition is also built double to resist pressure when one tank is full and the other empty. This partition, as well as the top of the studding of the outside walls, is covered with a 2 × 10-in. plank to serve as a walk. Usually two tie-rods run across in each direction in each half to take up side pressure.

The feed is admitted at the deeper end of the tank over the lip of a launder. By means of a gate the feed is diverted to either compartment. In the best construction, this launder is extended 10 in. or a foot toward the center of the tank, sufficient to admit a small vertical bottomless box extending almost down to the discharge spigot so that if it becomes clogged, it can be punched out easily. The clarified water discharges at the shallow end of the tank into a launder which conveys it to the settling pond. The rear side is made about four inches lower than the

other sides to prevent an overflow on all sides if the overflow launder should clog.

Water under pressure can be used for sluicing the settled material, but it is better practice to sluice with clarified water from the other half of the tank. A 1-in. pipe through the central partition connects the two compartments and by attaching a hose to this, the clear water from the tank which is being filled can be conducted to the far corner of the tank which is to be sluiced. It is guided along the periphery of the tank from this point by cutting a trench in the settled sands around the outer edge to a point over the discharge spigot. The current hugs the box until it reaches the bottom of the sand and then begins to eat toward the center, furnishing a product of constant thickness. A double tank of approximately the dimensions given is capable of handling about 250 tons in 10 hr. The dimensions are those used by J. G. Marcum in designing Joplin mills.

Classifier for Use Before Concentrators (By E. W. Durfee).—The desirability of close-screen sizing of mill products before concentration cannot be questioned by those familiar with the work. Could it be carried successfully to the fine sizes without too much expense, little would be desired in the classification of ores for concentration purposes. It is my opinion, however, that for sizes smaller than about 30 mesh the products from water classification will give equally good, if not better results for table concentration.

If one will observe closely the operation of a Wilfley table, handling the product from poor screen sizing (which is the rule in fine screening) he will notice that the finest particles of the heaviest mineral occupy the position closest to the table top and fill the interstices between the larger heavy particles. Next above are the lighter minerals similarly arranged and finally the top stratum is made up of coarse and fine quartz particles. In the operation of the table there is little trouble in washing off the coarse quartz particles, but there is always a certain amount of fine quartz lodged among the coarser, heavier minerals, which it is impossible to separate on the table. With these conditions in mind it is plain to see that water classification is the ideal system for making products for table concentration. The finer heavy minerals will go with the coarser lighter ones in which case the bedding on the table top will be, the finer heavier minerals, above which will come coarser, lighter ones that can be easily washed off, there being no spaces among the heavy mineral particles large enough to inclose and hold them back.

In making tests for the separation of pyrite and sphalerite in the complex lead, zinc, iron ores treated at the Daly-Judge mill in Park City, Utah, while general superintendent for that company, a novel classifier was tried and afterward installed on account of the excellent work per-

formed. The principle of the machine is based upon the free setting of minerals in water and the separation of the various products by revolving the columns of water in which the minerals settle around the central axis in a tank, thus distributing the classified products into different compartments. The products from these compartments are delivered through spigots to the concentrating machines.

The machine consists of a tank about 5 ft. in height by 4 ft. in diameter, the lower 3½ ft. of which is made so that it can be divided radially by sliding partitions. The number of these partitions depends upon the number of sizes required while their position can be determined by experimenting

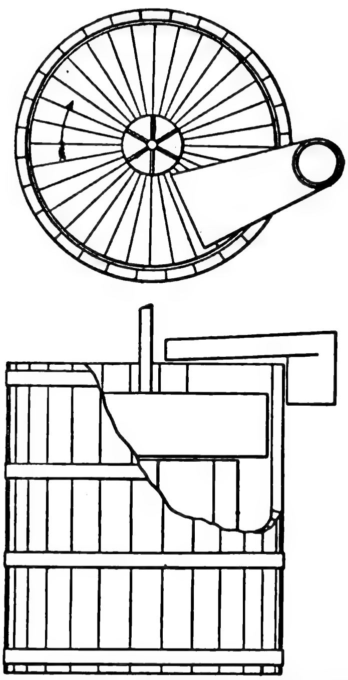


FIG. 80.—DALY-JUDGE CLASSIFIER FOR FINE MATERIAL.

with the product to be treated. The part that comprises the settling columns for the minerals revolves in a vertical position just above the partitions which separate the various sizes. It is made about 1 ft. long, of sheet iron, cylindrical in form and as large as will revolve in the tank, and has a central cylindrical core about 1 ft. in diameter, the periphery being divided into 32 equal compartments which are open at both ends. This revolves just below the surface of the water in the tank, making about 1 r.p.m. The feed is delivered from a stationary position at one side. As a compartment of the revolving unit passes underneath the feed, a portion is caught and settles as it revolves around the central

axis. The coarsest and heaviest particles passing through most quickly drop into the first compartment, the finer into the next, and so on around. The time of revolution can be adjusted so that the finest mineral will have settled through the revolving unit by the time the compartment has returned to the position underneath the feed. Fig. 80 shows a plan view and side elevation and partial section of the assembled machine.

Increasing the Effectiveness of Spitzkasten[1] (By Beauchamp L. Gardiner).—The ore from the Sons of Gwalia mine is mainly schistose in character, and consequently the stamp milling and subsequent fine grinding in pans produce a relatively large percentage of slime (50 %), of which a great portion is colloidal and slow settling. The separation of the sand from the slime at that mine is effected primarily by a row of five small spitzkasten, each 3 × 3 × 3 ft.; the underflow goes to sand-collecting vats provided with Butters distributers and annular launders, which effect a further separation, while the slimy overflow joins that from the classifiers and is delivered thence to the slime-treatment plant. This product, which is termed slime, is characteristic of the slime at most Western Australia plants and contains nearly all the finely reduced colloids together with a quantity of fine crystalloids or sand. It is slow settling, and requires an extensive plant to thicken it sufficiently for economical cyanide treatment. The inflowing stream, which contains the equivalent of 6 to 8% of dry slime, first delivers to a nest of spitzkasten 30 × 40 ft. The underflow from these is drawn off through 1-in. nipples (into which wooden plugs with holes bored through them are inserted) and by substituting plugs with larger or smaller bore, as the case may require, it is regulated so as to keep the overflow just running clear. The underflow, which ranges from 14 to 18% of dry slime, delivers to another nest of spitzkasten 32 × 24 ft., operated in a manner similar to the first, and the pulp is here thickened to from 23 to 27% (a clear overflow being maintained), and then it gravitates to two dewatering sumps working on the vacuum-filter principle, where it is finally brought up to from 40 to 45%. The quantity of slime handled ranges from 150 to 200 tons per day.

At the Sons of Gwalia cyanide plant the pulp-thickening process is therefore of some moment and any means of accelerate the rate to settlement would be of great advantage. Experiments have been made along lines which practically mean the creation of additional area in the existing arrangements. This is done by suspending in the pulp, in an inclined position, a number of sheets of corrugated iron and it has been found that these sheets materially increase the rate of settlement, thereby making it possible to obtain a thicker pulp.

Before giving the details, consideration of the following experiments will

[1] Excerpt from an article in the *Monthly Journal* of the Chamber of Mines of Western Australia, entitled "Slime Settlement."

explain the principle involved: A glass cylinder of slime is set up in a vertical position and the slime allowed to settle; the rate of settlement is 40 mm. per min. The cylinder is again agitated and then placed at an angle of 45 deg. from the vertical, as shown in Fig. 1, referring to Fig. 81; the rate of settling is then 180 mm. per min., or 4½ times as fast. The reason for this is quite obvious on close examination, for, on the upper side of the cylinder, a stream of clear water can be seen flowing upward and joining

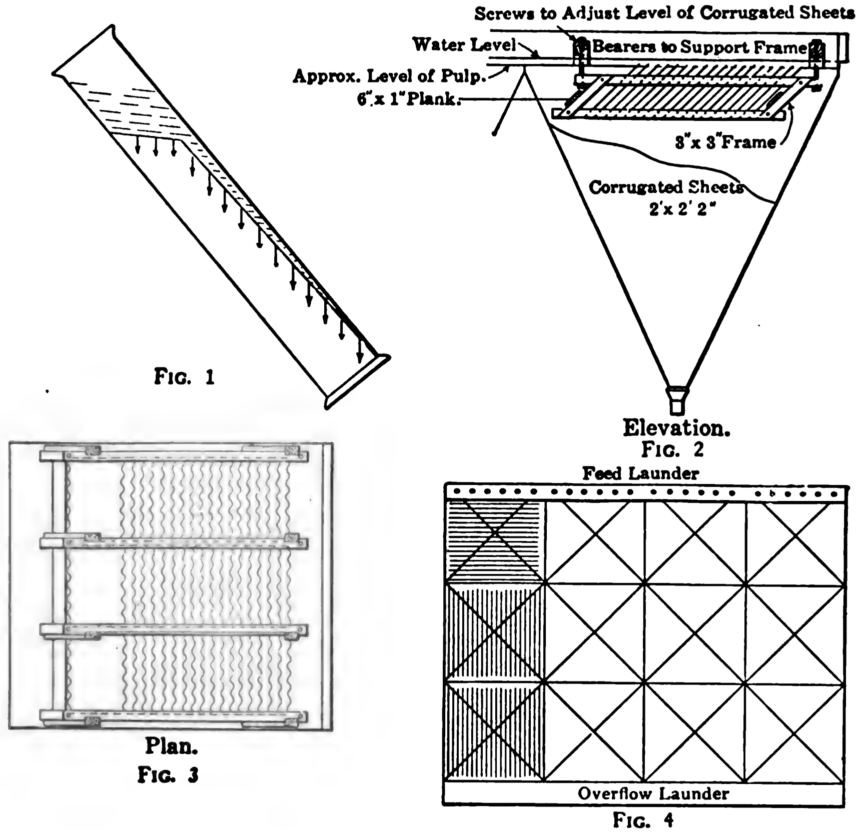


FIG. 81.—USE OF CORRUGATED PLATES IN SPITZKASTEN.

the supernatant clear water above the pulp. Evidently the particles of slime settling in a vertical direction, as indicated by the arrows in Fig. 1, must leave, upon any part of the tube facing downward, a layer of clear water which is immediately displaced upward owing to its lesser density compared with the pulp that moves downward to take the place of the displaced water. The use of the inclined sheets of corrugated iron is an application of this principle; wherever there is a surface facing downward, the pulp must settle away from it, leaving the water to be forced up. The

corrugations deflect the flow into narrow streams, which break the surface at points and do not disturb the natural settling tendency of the slime.

In Figs. 2 and 3 (Fig. 81) is shown the manner of suspending the sheet, which experience has proved to be most effective. Three spitzkasten were isolated from the rest, as shown in Fig. 4, and each was provided with inclined sheets. Where possible it is best to set them parallel with the current, so as to impede the flow of pulp to the least extent. In the present case, the sheet in the first spitzkasten had to be set crosswise in order to clear the feed launder, but as the flow is not great in that part, it makes no difference. The top edges of the sheet are set just flush with the surface and by regulating with plugs in the underflow, the pulp level is maintained at from 2 to 3 in. below the surface. Small streams of water flow up each of the corrugations to join the clear water above and the slime tends always to maintain its hydrostatic level—the demarcation of slime and water being a horizontal plane over the whole area—with the sheets of iron projecting through it to a greater or less extent, according as the pulp is high or low. The natural settling area is scarcely interfered with, the small jet of water in breaking through affecting it only to a slight extent, so that there is practically the original settling area plus the total area of the sheets multiplied by the cosine of the angle they make with the vertical, or the projection of their area on the horizontal plane.

TABLE XXIV.—COMPARISON OF SPITZKASTEN WITH AND WITHOUT ACCELERATING DEVICE

No. of test	Spitzkasten 24 × 8 ft. = 192 sq. ft., with settling device		Spitzkasten 24 × 24 ft. = 576 sq. ft., without settling device	
	Pounds settled per minute	Percentage of dry slime	Pounds settled per minute	Percentage of dry slime
1	47.3	42.6	168.0	25.0
2	50.4	40.0	139.3	24.4
3	40.6	40.9	144.3	27.2
4	54.4	41.7	144.5	27.2
5	53.0	40.0	182.0	26.6
Average.........	49.1	41.1	155.6	26.1

The figures given in Table XXIV show the efficiencies of the ordinary spitzkasten compared with those fitted with the accelerating device. These tests were all made on slime resulting from sulphide ore, the inflowing pulp being from 15 to 18% of the weight of the ore. The conditions were, that a clear-water overflow had to be maintained and that the pulp level had to be stationary at 2 to 3 in. under the surface and maintained in that position for at least half an hour before any measurements were made. Endeavor was always made to have an amount of pulp propor-

tional to the areas running into each section. The figures in the table were obtained by catching the underflow in a receptacle for a measured time, and by weighing the amount caught; from this and the density, which was also determined, the pounds per minute of dry slime were calculated. The averages show that the feed is nearly proportional to the areas, so that the net result is to increase the thickness of the pulp from 26 to 41%. It is proposed to fit the whole of the nest of spitzkasten with these accelerators and it is expected that the present mechanical dewatering plant will be wholly or partly dispensed with.

The area of the experimental spitzkasten is 192 sq. ft., and the total area of the inclined sheets used is 648 sq. ft.; they are inclined at 45 deg., so that the effective area is 450 sq. ft. Taking the inflow at the average value of 17% for both columns shown in the accompanying table, and calculating the amount of water removed, then with the figure so obtained divided by the area, the fall in inches per minute can be determined.

For the unaltered spitzkasten the fall is 0.107 in. per min.; for the spitzkasten divided with accelerators the fall is 0.168 in. per min. The increase in the settling rate is not proportional to the increase in effective area. The reason for the difference lies in the fact that, of the sheets dipping some 18 in. into the pulp, the bottom portions are surrounded by a mixture of greater density than the surface layer and consequently there is less settlement per unit area for those portions; moreover, the small uprising streams of water mix again to a certain extent with the pulp, and this tends to decrease the efficiency.

The chief merit of the device is that it can be applied to existing spitzkasten at small cost, and that a given amount of settling can be performed in smaller space. It may be used for increasing the thickness of the pulp or enabling larger quantities of pulp to be handled and should be effective if applied to the decantation process, the greater rate of settlement increasing the capacity of the plant considerably.

The principle certainly has its limitations. Depending as it does upon the behavior of the slime and water as two immiscible fluids, its effect is not so marked on thin pulp—the thin seeming to mix more easily than the thick pulp—and the difference in density is not so noticeable. In all cases, the simple laboratory test of the inclined glass cylinder gives a good idea of the possibilities. By determining the settling rate in a vertical position and then inclining at different angles, calculating the increased effective area at the settling rate, a good idea can be formed of the results likely to be obtained on the larger scale.

Cast-iron Spigot Holder for Settling Cone.—To avoid the flooding usually experienced in changing spigots to cone settlers, W. O. Borcherdt, assistant superintendent for the Bertha Mineral Co., at Austinville, Va., devised the spigot holder shown in Fig. 82. The holder is pivoted at the

$\frac{3}{4}$-in. drilled hole, and spigots are screwed into both the $1\frac{1}{2}$-in. holes. In case it becomes necessary to change or renew a spigot the bolts can be loosened and the other spigot simply swung into position while a new one is being put into the holder. Thus, instead of exposing a much larger aperture and probably disarranging the working of the entire mill, a spigot of the proper size is always at hand if the one in use becomes clogged or requires attention for any cause.

Drag Classifiers

Esperanza-Federal Classifier (By Frederick MacCoy).—The Esperanza-Federal classifier, which is the joint invention of H. A. Guess,

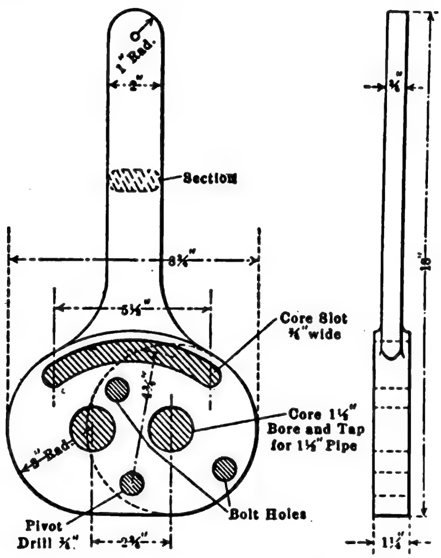


FIG. 82.—CAST-IRON SPIGOT HOLDER.

manager of the Federal Lead Co., Flat River, Mo., and Charles Hoyle, manager of the Esperanza Mining Co., of El Oro, Mex., is used in the El Oro district for separating the slime from the sand before tube milling. At Esperanza two batteries of these classifiers are used. The upper battery takes the stamp-mill product and separates the flocculent slime from the quartz sand, and sends the sand to the tube-mill plant for regrinding, and the slime to the upper Pachucas. The lower battery is working on the tailings dump which was built up in the days before the tube-mill plant was installed. This tailings dump contains a greater proportion of sand, as much of the original slime has been washed away.

The essential features of the classifier, to quote the description of

same on which U. S. patent No. 565,801 was based, are: "The combination of a settling box having an inclined bottom extending above the overflow level, an extended overflow pan in said box spaced from the ends thereof and presenting an overflow edge, a conveyor in said box encircling the overflow pan in a vertical plane and comprising an endless series of transverse scrapers extending across said box and adapted to engage the bottom, said series extending along said bottom to above the liquid level."

As will be seen by Fig. 83, the sand and slime enter the classifier through the feed launder; the lighter slime overflowing into the slime-discharge pan and from there out through the slime-discharge pipe. The sand, being heavier, settles to the bottom of the box, where it is caught by the scrapers and elevated above the overflow level and discharged over the top of the inclined bed of the classifier.

Various ratios of classification are possible, by altering the speed of the drag chain, spacing of the drags, inclination of the classifier, varying

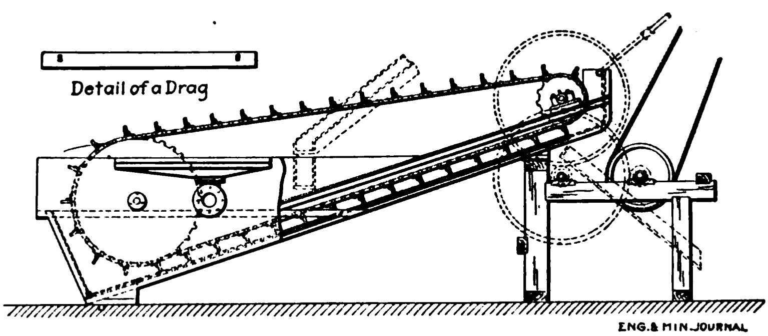


FIG. 83.—DRAG CLASSIFIER USED IN EL ORO DISTRICT, MEXICO.

height of overflow slime pan, sprinkling of the sand as it is elevated above the overflow, etc.

Various materials have been used at Esperanza for the scraper bars, such as slotted trommel plates, solid iron bars and angle irons, but the most satisfactory material has been found to be ordinary wooden slats, as they keep ground down to conformity to the bottom, and are inexpensive to replace when worn out. The metal bars develop a tendency to arch somewhat from continued use, and thereafter do not engage the bottom as well as do the wooden bars.

At the mill of the Mexico Mines of El Oro, Ltd., the classifier is made use of as a combined classifier and oversize return on the tube mills. A classifier is placed alongside each tube mill and receives the combined discharge from the stamp batteries and the tube mills. The slime is cut out, and the oversize from both sources is elevated by the drags to a point high enough above the feed end of the tube mill to flow in by gravity.

The classifier has been installed in all of the mills in the El Oro district, with slight variations in dimensions, etc., from the original as built at Esperanza. As indicative of the work which the classifier does, a resumé of tests under working conditions is given in Table XXV. Samples were taken every 30 minutes.

TABLE XXV.—TESTS OF CLASSIFIER AT ESPERANZA MILL

Test number	1	2	3
Tons treated	11,056	4,653	17,035
Number of classifiers	3	2	2
Screen test on feed	%	%	%
+ 40 mesh	7.4	5.3	1.9
+ 60 mesh	9.8	10.1	8.5
+ 80 mesh	8.3	10.0	9.2
+100 mesh	1.4	1.9	1.2
+150 mesh	21.4	23.0	23.9
+200 mesh	1.6	0.8	0.4
−200 mesh	50.1	48.9	54.9
Screen test on sand product:			
+ 40 mesh	15.1	15.3	9.1
+ 60 mesh	20.2	24.2	25.2
+ 80 mesh	15.9	18.4	20.6
+100 mesh	2.6	2.8	2.4
+150 mesh	31.3	30.0	32.4
+200 mesh	1.9	0.7	0.4
−200 mesh	13.0	8.6	9.9
Screen test of slime product:			
+100 mesh	0.5	0.8	0.9
+150 mesh	5.7	6.9	8.8
+200 mesh	1.0	0.5	0.4
−200 mesh	92.8	91.8	89.9

Increased efficiency in concentration at the Federal mill, Flat River, Mo., is attributed to the use of Esperanza-Federal classifiers because they give a clean separation of sand and slime and also furnish an even table feed by acting as accumulators.

The Major Classifier.—The Major classifier, used in the Minnesota mill at Maitland, S. D., consists of a cylinder or drum mounted upon rollers so that it can be revolved, and having a bevel-gear arrangement for applying the power for rotation. The ends of the cylinder are partly closed, leaving openings varying in size with the requirements of the machine. The cylinder is divided in two portions by a partition, the height of which is somewhat less than the height of the end walls of the cylinder. Inside the cylinder are longitudinal ribs or blades fastened to the drum, these blades forming pockets or gutters with the cylinder shell.

The operation of the machine is as follows: The pulp is introduced into

the drum, revolving at from 1½ to 2 r.p.m., by means of a launder which enters through the overflow end of the cylinder and delivers the pulp at the opposite end of the drum. The launder is shown at *A*, Fig. 84, and at *B* is shown the partition which divides the interior into two parts. The blades, *C*, tend to agitate the pulp, and the lighter portion is overflowed past the partition wall into the second compartment; the coarse sand settling in the pockets of the blades is elevated as the drum revolves and discharged with the assistance of a water or solution spray into a launder conveniently placed to receive it. This launder is shown at *D*, as are also the pipe and spray connections, *F* and *G*, for dislodging the coarse sand elevated in the pockets.

The portion of pulp which overflows into the second compartment repeats the same process exactly, except that the material is finer, and there may be a different arrangement of the blades or ribs with the object of taking out a product finer than that extracted by the primary compart-

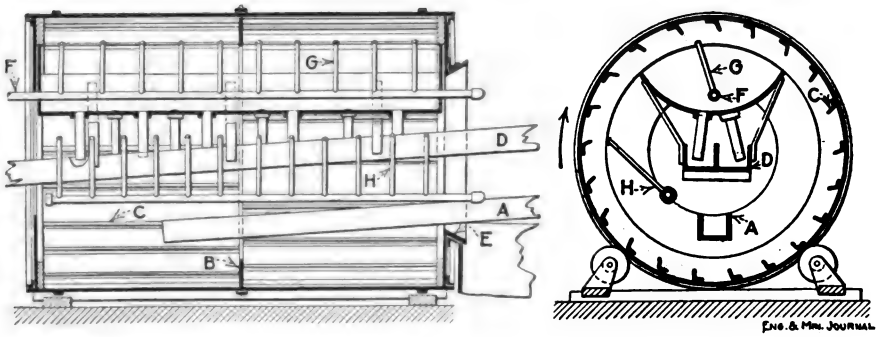


FIG. 84.—MAJOR CLASSIFIER USED AT MINNESOTA MILL, BLACK HILLS.

ment and leaving a pure slime product to be overflowed from the machine at the rear end *E*, this wall being lower than the wall partially closing the opposite end of the machine.

As the sand and middling are carried upward by the blades in their respective compartments, they are washed by a spray *H*, after which the discharging spray washes the material off into the proper launder or tray. In order that there may be two distinct products kept separate, the discharging launder is usually divided into two parts, or made as two launders running side by side, arrangement being made to dump each class into its particular launder. These launders may be arranged to discharge in different directions if it is so desired, but it has been found better practice to have them discharge at the same end, the one opposite to that through which the pulp enters. The drawings show the arrangement and operation of the machine.

The particular advantages obtained by the use of the machine are a

saving of almost the entire fall, as the machine acts as an elevator as well as a classifier, and it occupies less floor space than the usual mechanical devices used for the purpose. The power consumed is small and the machine has a large capacity. A machine 6 ft. in diameter and 7½ ft. long, when handling 30-mesh and finer solids at a ratio of one of solids to seven or eight of liquid, has a capacity of about 150 tons of solids per 24 hr. Three products can be obtained, a coarse sand for regrinding, a finer sand, or middling, for leaching, and a slime for agitating, or any other convenient utilization of these products. If it were considered desirable, the machine could be easily developed to deliver additional products as required. No attention is necessary, as the machine works efficiently with varying feeds up to its maximum capacity. In case of a forced

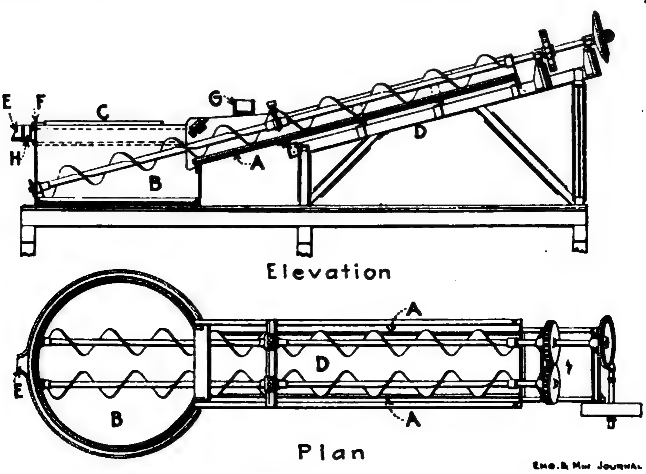


FIG. 85.—ARGALL SPIRAL CLASSIFIER.

stoppage of the mill, the machine does not choke up and can be easily started even if filled with pulp.

The machine installed at the Minnesota mill is 6½ ft. in diameter and 8 ft. long and handled over 200 tons of solids per day with ease. Its cost is low and maintenance is slight. It was designed by Edmund Major, superintendent of the Minnesota mill, and has been patented in the United States, Canada and Mexico.

Argall Classifier.—Philip Argall, of Denver, Colo., was granted a patent (U. S. No. 1,044,844) on an apparatus for separating sand and slime. The object is accomplished by means of oppositely rotating spiral conveyors operating within an inclined trough. The sand is discharged at the upper end of the trough, while the water and slime flow over a weir at the lower end of the trough. Classifiers of this general type have been employed successfully at Stratton's Independence mill,

Victor, Colo. In a second patent (U. S. No. 1,044,845) Mr. Argall likewise uses spiral conveyors intergeared to rotate in opposite directions and further provides an especially long weir over which the suspended slime is flowed from the settling portion, so that slime of great fineness may be obtained. Provision is also made for adjusting the length of the weir to obtain slime of any desired degree of fineness.

The conveyors are adapted to be rotated slowly, conveniently from three to eight revolutions per minute. A small space is allowed between the conveyor and the bottom of the trough so that the sand may build up and form a bed. On the bottom of the trough *D*, referring to Fig. 85, and extending the full length thereof, are secured a pair of angle irons *A* which are placed just outside of the edges of the conveyors. These angles are of relatively short height to constitute, with the side walls of the trough, overflow channels. The feed launder *G* is placed at sufficient distance from the connection of the trough *D* with the settling tank *B*, so that the heavy sand will not enter the settling tank but will be pushed upward by the screw conveyor as fast as it falls thereon, thus leaving the tank for the free settling of the finest sand. The operation of the conveyors displaces the sand particles, causing the slime to be squeezed out of the sand and to be returned to the settling tank by the overflow channels mentioned above. The slime-freed sand is discharged from the upper end of the trough.

Seated within the channel between the wall of the overflow launder *H* and the settling tank *B*, and extending for the full length thereof, is a weir *F*, the upper edge of which is level and projects above the upper edges of the settling tank and overflow launder. By means of the dam *C*, which is preferably formed of a number of overlapping sections, any portion or all of the weir *F* may be cut out of operation. If the dam is not utilized, the entire length of the weir will be effective, thus providing a long weir, which is highly desirable when a slime of fine mesh is to be obtained. By varying the length of the weir and by changing the speed at which the conveyors are driven, the machine can be adjusted to give slime of any desired degree of fineness. It is also noteworthy that, by reason of the substantially circular form of weir employed, a weir of great length is obtained in a comparatively small area. The overflow launder is provided with a suitable discharge opening *E* through which the slime is discharged.

Regulating Moisture in Sand Discharged by Dorr Classifier (M. G. F. Söhnlein).—If thin pulp is fed to a Dorr classifier, the sand discharged often contains an objectionable amount of moisture. This can be remedied by a simple arrangement illustrated in Fig. 86, in which the rake *R* is at the end of its upward stroke. A ⅛-in. iron plate, which has the same width as the classifier, is bent into the shape *A B C* and bolted

to the bottom at the discharge end. The sand is piled up in *A B C* and loses much water, before it is discharged over the lip *B C*. Every time the rakes bring some wet sand up to *A*, an equal amount of dryer sand leaves the classifier at *C*. I have made *A B* = 5 in. and *B C* = 2 in., but these measurements have to be varied according to the quantity of sand and the desired degree of moisture. By using this appliance the amount of water in sands could be reduced from 60% to only 12%. I conceived the idea by watching a classifier work when a bank of frozen pulp had

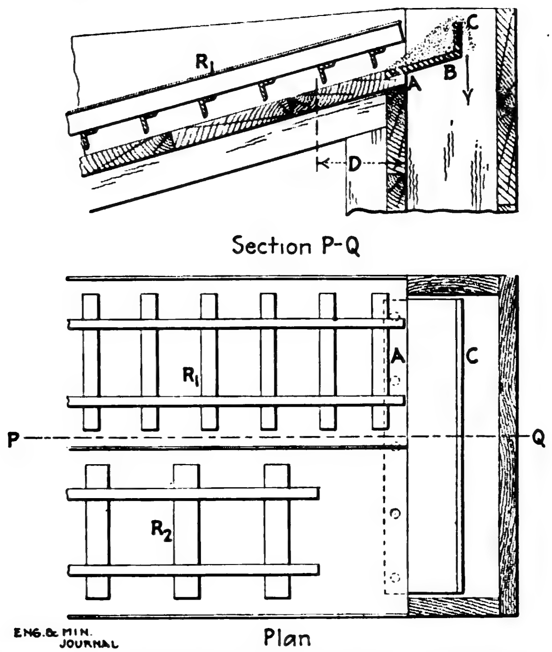


FIG. 86.—MOISTURE REGULATOR ON DORR CLASSIFIER.

been formed at the discharge end, which had almost the same effect as the lip.

[The Dorr classifier, when required, is equipped with a device for accomplishing the same result. At the end of the rake travel, for any convenient distance *D*, in the drawing, a filter bottom is inserted, which may be connected with vacuum or not, as required. The rake-travel bed may be made longer to allow for this installation if necessary.—EDITOR.]

Automatic Indicator for Drag Classifiers (By Donald F. Irvin).—The harmful effect upon the efficiency of drag classifiers of variations in the solid to liquid ratio of the pulp fed was clearly demonstrated at the mill of the Tigre Mining Co., El Tigre, Sonora, Mex., during the winter of 1911–1912. At this mill five drag classifiers of the Esperanza type are

used. Two of these handle the current mill pulp, while the other three are used in classifying the pulp discharged from the five tube mills.

When the mill was put in operation, a ratio of solution to ore was fixed. Having speeded shafting and pumps, and regulated valves accordingly, the classifiers handling the tube-mill discharge were assumed to be operating under the desired conditions. Work had not progressed far when it was found that temporary variations in the amount of dry pulp delivered to the classifier were sufficient to upset the requisite conditions for good classification and at times the classifier was sending fine sand into the slime overflow. These variations in feed were caused most often by irregular feed of dry sand from the sand-tailing reclaiming plant; from 50 to 75 tons of old concentrator sand tailing are delivered to the tube-mill department daily.

To remedy this condition, the spray pipe at the head of the classifier, which is used to dislodge any sand that sticks to the rakes, was opened wider. This caused the spray to shoot over into the body of the classifier where it acted as a diluent. At the same time enough solution dropped into the discharge launder to carry the sand to the tube mill. This

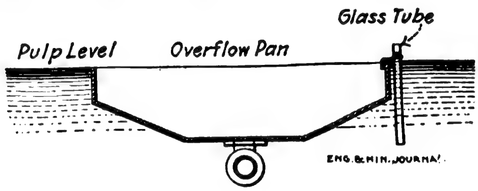


FIG. 87.—PULP INDICATOR FOR DRAG CLASSIFIER.

expedient helped classification, but if carried to excess so that the slime discharge was truly slime, too much solution was sent to the Dorr thickener. This method of regulation was obviously purely experimental and was trying to the operator of the plant. Recourse to hydrometric measurements was not to be thought of, as it would involve the use of fragile, high-priced hydrometers by ignorant, low-priced Mexican laborers. Furthermore, the necessity of making frequent hydrometric measurements was to be avoided if possible.

A float hydrometer described by Charles F. Spaulding in the *Mining and Scientific Press* for use in agitating tanks, suggested the indicator which was worked out for the drag classifiers at Tigre mill. The device of Mr. Spaulding consisted of a piece of pipe 10 ft. long, equipped with a float and vertical scale. The hydrometers were left floating in the tank so that the thickness of the pulp, as evidenced by the reading on the scale, could be told at a glance.

The device used at the Tigre mill consists of a piece of ⅝-in. gage glass 10 in. long, which is suspended from the edge of the slime-overflow pan

by means of a bent wire as shown in Fig. 87. When put into place the level of pulp inside the glass tube is naturally the same as that on the outside, but in a few minutes the meniscus begins to rise, the contents of the tube become clearer, showing that there has been a settling out of the solid particles, and a corresponding decrease in specific gravity; buoyed up by the heavier pulp outside of the tube, the level of the meniscus rises.

In practice, these tubes show the meniscus about ¼ to ⅜ in. below the top of the tube, and about 1¾ in. from the meniscus to the level of the pulp. This position for the meniscus was chosen by experiment as the point at which, on El Tigre ore, the thickness of the slime overflow was such as to insure 90% of it passing a 200-mesh screen. If the supply of solution for the classifier decreases, or the supply of ore increases unduly, the pulp becomes thicker, its specific gravity becomes greater, the internal column in the glass tube rises higher and higher, and even may steadily overflow the top of the tube. Should the ore supply slacken, or the solution supply increase, the internal column will, on the contrary, lower and tend to coincide with the level of the pulp outside.

The guide, therefore, to the proper running of the classifiers is extremely simple and can be condensed into the statement that 90% of the slime overflow will pass 200 mesh when the meniscus is ¼ in. below the top of the tube. Seldom does there occur a large excess of solution, as with increased tonnage the pump capacity of the cyanide plant can hardly do more than keep the proper quantity of solution in circulation. A liquid to solid ratio of 5 : 1 or 5½ : 1 in the classifiers insures the specified degree of sizing.

The setting of the tubes was done indirectly, by first varying the pulp thickness and observing the liquid to solid ratio, and the percentage of 200-mesh material in the slime overflow. Finding that 5 or 5.5 : 1 gave the desired results, the tube was measured for height of water column at those figures, and a datum level established for regular work.

These indicators are not used on the two concentrator-tailing classifiers as there is always a large amount of water in them and classification is uniformly good. This device is offered to the cyaniding community in the belief that it has not been used elsewhere for this purpose.

Overload Alarm for Dorr Thickeners.—An interesting and ingenious device has been arranged to be placed on Dorr thickener tanks to call attention to an overloaded condition. The arrangement, as illustrated in Fig. 88, is as follows: Above the usual gear wheel, which moves the vertical shaft, is placed a cast-iron arm keyed to the vertical shaft. The gear wheel itself is not keyed to the shaft, but is loose on it and free to move independently. The arm and gear wheel are connected by means of a spring, the tension of which may be adjusted for working conditions. The power being applied, as usual, to the gear wheel, the arm is moved through

the spring, and the vertical shaft moves with the arm. However, should the load be too heavy, the gear wheel continues to move but the arm remains stationary, the load on the shaft being too much for the spring, which opens. Hence the relative position of the arm and gear wheel is not maintained and the arm drags back. When the arm has moved, with reference to the gear wheel, a few inches, it presses on a small plunger rod, which makes an electrical connection and rings a bell. The connection and bell are fixed on the large gear wheel, and the movement of the arm need be only small to make the connection and ring the bell. Attention is thus immediately called to the condition of the thickener and the mill man takes immediate steps to lighten the load by cutting off the feed to

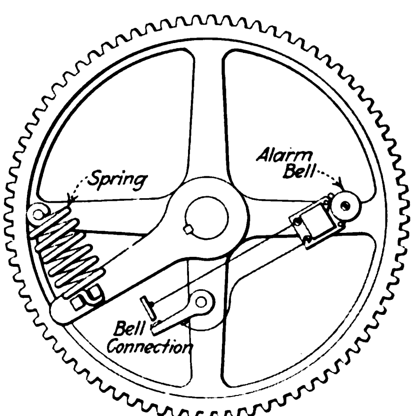


FIG. 88.—OVERLOAD ALARM ON DORR THICKENER.

the machine or increasing the discharge or any other convenient method. The contrivance works well and is of great use on thickeners, particularly where the feed or discharge to them is variable. It is used on the Dorr thickeners at the mill of the Blue Flag Gold Mining Co., of Cripple Creek.

JIGS

Notes on Operation

Jigging Practice in the Coeur d'Alene.—The Federal company is said to be obtaining satisfactory results from the Hancock jig on the finer sizes of ore. This is not surprising, inasmuch as experience in the Coeur d'Alene has shown that the Harz jig, as constructed and operated there, is susceptible of great improvement. The Bunker Hill & Sullivan company has achieved such improvement, particularly with the finer sizes, by the classifying Caetani jig, which is considered to do just as good work as the Hancock, while it has the additional advantage of successfully separating

all the slimes and sending them to the slime department, directly from the first compartment, without causing them to travel the length of the jig, as the Hancock arrangement demands. The old-style Harz jig will probably, in the course of a few years, be entirely superseded in the Coeur d'Alene by improved jigs of the type of the Hancock and Caetani.

Joplin Jigging Practice (By Claude T. Rice).—The roughing jigs are 5- or 6-compartment machines, although some 4- and a few 7-compartment jigs are in operation. The 7-compartment jigs are used in mills wherein the ore treated contains much lead or the jig feed is rich. The grate area of each compartment or cell ranges from 30 × 42 in. to 36 × 48 in. The jig shafts make 90 to 110 r.p.m.; the stroke ranges from 5/8 to 1 in. In a 6-cell jig the extreme difference of stroke in the different compartments is 1/2 in. The bed of material on the grates is usually 5 in. thick; in some instances the thickness is 4 in., in others 6 in. The roughing jigs are operated with the hutch gates partly open, which tends to increase the strength of the suction stroke. There is a drop of about 3 1/2 in. from one compartment to the next, and the grates are set at an inclination such that the discharge is about 1 in. lower than the upper end. The cleaning jigs are 6- or 7-compartment machines; usually there is one more cell in the cleaning than in the roughing jigs. The grate area of each compartment ranges from 24 × 36 in. to 36 × 48 in., 30 × 42-in. grates being the most common size. The bed of material is 4 to 5 in. thick; the shafts make 135 to 160 r.p.m., and the stroke ranges from 1/4 to 5/8 in. The drop from one compartment to the next ranges from 1 5/8 to 1 3/4 in. The grates are set at an inclination, as in the roughing jigs. The sand jigs are 4- or 5-compartment machines; the grate area ranges from 20 × 30 in. to 30 × 42 in., 24 × 36 in. being the most common size. The thickness of the beds is usually 5 in.; the shafts make 150 to 190 r.p.m., and the stroke ranges from 1/8 to 3/8 in., 1/2 in. being the stroke usually preferred. The drop between cells is 1 inch.

Although the practice of setting the grates at an inclination results in unequal working of the current, because of the thick beds there is still enough thickness of zinc-ore particles at the head of the grate to clean the product going into the hutch. The beds of both roughing and cleaning jigs are supported by wooden or cast-iron grates, which are placed so that the direction of the slots is at right angles to the flow of pulp. The slots in the grates of the roughing jigs are 1/8 in. wide, in the grates of the cleaning jigs 1/12 in. wide, and in the sand jigs 1/12 to 1/16 in. wide. The feed to the roughing jigs contains an excessive amount of water. Hutch products are drawn off through gates that are kept partly open, so that there is a uniform flow of the products from the jig compartments. The chats are drawn continuously through special gates from the bedded material on grates.

The capacity of a 36 × 48-in., 6-compartment roughing jig is 250 tons per 10-hr. shift. The capacity of a 30 × 42-in., 7-compartment cleaning jig is 75 to 100 tons. From 1500 to 2500 gal. of water is used per ton of ore treated. A large proportion of concentrates passing 2-mm. openings is recovered by the jigs. This product comes from the cleaning as well as from the sand jigs, because much blende that will pass 1-mm. openings is drawn into the hutches by the strong suction.

Bull-jig Rougher in a Joplin Zinc Mill (By L. L. Wittich).—To abolish the necessity of regrinding much of the zinc ore that would be returned to the rolls after failing to pass through a 5/8-in. revolving screen, the Culpeper Mining Co., north of Carterville, Mo., has installed what is termed a bull-jig rougher, and claims the capacity of the mill is increased 100 tons per day, making its total capacity 500 tons. This is the first instance of a rougher jig of different construction from the regular rougher being installed in the Joplin district. It is situated high in the building and handles only the coarse ore that passes through a 1/2-in. revolving screen. The regular rougher cares for the ore passing through the regulation 3/8-in. mesh. The operators claim the extra cost of running the bull-jig is too insignificant to be considered. The bull-jig rougher is composed of two large cells, each 36 in. wide and 60 in. long. Four eccentrics, two for each cell, operate the jig beds. Were it not for the bull-jig rougher a large volume of chats, on failing to pass through the 3/8-in. screen would be returned to the rolls for regrinding. Of the concentrates, neither draw-off nor hutch product from either cell, go directly to the bins. A return of this mineral-bearing ore is made to the chat rolls where it is reground. Ore that passes through the 3/8-in. screen then goes to the regulation rougher, from the first cell of which the draw-off goes directly to the bins. Tailings from both the regulation rougher jig and the cleaner, are drained over revolving screens, the water passing into settling tanks from which the slime feed is drawn for the tables.

Designs and Construction

Joplin Hand Jig.—The details of the design of the hand jigs used in the Joplin district of Missouri are shown in Fig. 89, which is self-explanatory. In Table XXVI is given a bill of the materials required for building one of these hand jigs. In the first column of the table, letters are given referring to the corresponding parts on the drawing.

The sieves are approximately 5½ ft. long, 2 ft. wide and 10 in. deep. They are suspended from the jig pole by strap-iron swings, connected through a slot to the jig-pole frame by angle bolts, and when new have 1/4-in. play up and down, but as this link wears the play increases to 1/2 or 3/4 in. This is desirable as it makes a quick suction stroke and a slight pause between the pulsion and the suction strokes. The jig pole, which

is a 2 × 6-in. plank 20 ft. long, is provided with a bolt at the end to keep it from splitting. In jigging, the operator stands on a spring board and grasps the cross bolt that goes through the jig pole as a handle. This spring board is arranged so that the bolt comes about waist-high while he is jigging. Before beginning to jig, the bed is settled and evened by a

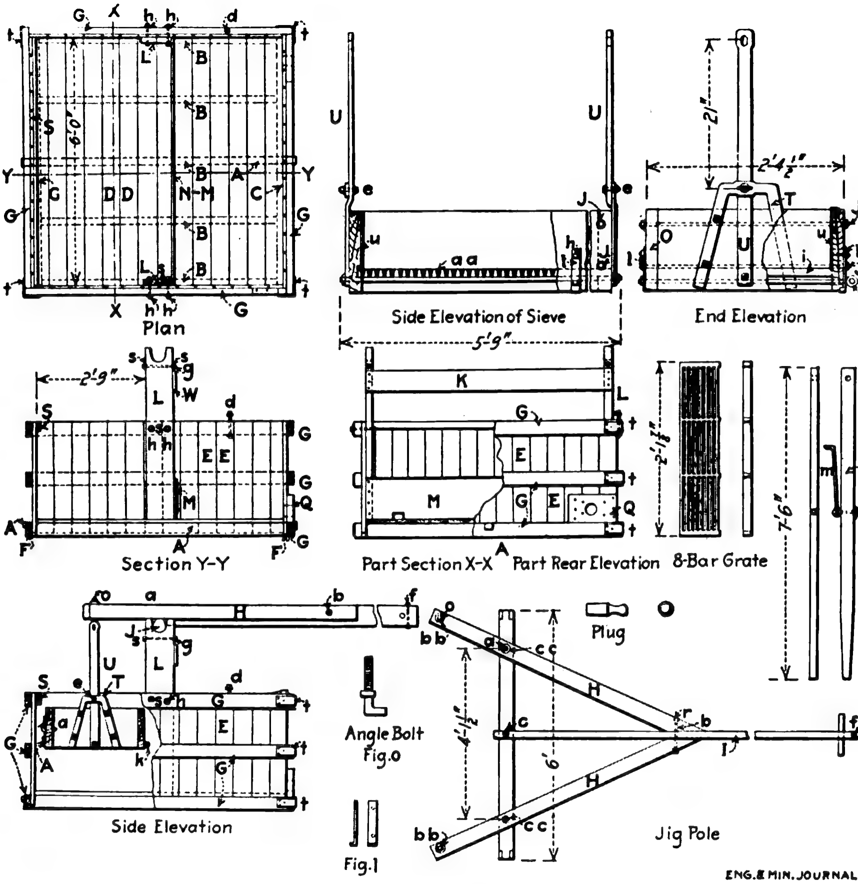


FIG. 89.—DETAILS OF A TYPICAL JOPLIN HAND JIG.

few vigorous up-and-down strokes with the sieve well submerged. Then the jigging proper begins with its peculiar short stroke, which is made much easier on the operator by the spring board.

The box of the jig is about 2 ft. 4 in. deep, 5½ ft. wide and 6 ft. long. The screen cradle is carried by two uprights that come up about in the middle of the side walls of the tank. The jig box is divided into two equal parts by a baffle-board about 12 in. high, so as to confine the hutch concentrate under the sieve while providing a quieter compartment for the settling of the finer sizes of concentrate that come through the

TABLE XXVI.—BILL OF MATERIAL FOR JOPLIN HAND JIG

Fig.	Number of pieces	Material	Fig.	Number of pieces	Material
A	1	2 × 4 in. × 6 ft. 6 in	f	1	$\frac{3}{8}$ × 6-in. bolt
B	4	2 × 4 in. × 5 ft. 8$\frac{1}{2}$ in	g	2	$\frac{3}{8}$ × 8-in. bolts
C	2	2 × 4 in. × 6 ft	h	4	$\frac{3}{8}$ × 4$\frac{1}{2}$-in. bolts
D	14	$\frac{7}{8}$ × 5$\frac{1}{4}$in. × 6 ft. 6 in. flooring	h_1	10	$\frac{3}{8}$ × 2-in. bolts
E	56	$\frac{7}{8}$ × 5$\frac{1}{4}$in. × 2 ft. 8 in. flooring	i	1	$\frac{5}{8}$ × 5 ft. 9-in. rod
F	4	$\frac{7}{8}$ × 1 in. × 6 ft. 2$\frac{7}{8}$ in	j	4	$\frac{5}{8}$ × 3$\frac{1}{2}$-in. rods
G	12	2 × 4 in. × 6 ft. 3$\frac{3}{4}$ in	k	1	$\frac{5}{8}$ × 31$\frac{1}{2}$-in. eye-rod
H	2	4 × 4 in. × 7 ft. 1$\frac{1}{2}$ in	l	2	1$\frac{1}{4}$ × $\frac{3}{8}$ × 6-in. irons
I	1	2 × 6 in. × 20 ft	m	1	$\frac{1}{2}$ × 19-in. eye-hook
J	1	4 × 6 in. × 6 ft	n	1	$\frac{1}{2}$ × 5-in. eye-bolt
K	1	2 × 4 in. × 7 ft. 6 in	o	2	$\frac{3}{4}$ × 2-in. angle bolts
L	2	2 × 8 in. × 4 ft. 2 in	p	5	$\frac{5}{8}$-in. cut washers
M	1	1 × 12 in. × 6 ft	r	8	$\frac{1}{2}$-in. cut washers
N	1	1 × 6 in. × 6 ft	s	14	$\frac{3}{8}$- n. cut washers
O	2	2 × 12 in. × 5 ft. 6 in	t	12	Corner irons
P	2	2 × 12 in. × 2 ft. 1 in	u	2	9$\frac{1}{2}$-in. × 2 ft. 1-in No. 16 sieve liners
Q	1	2 × 8 in. × 14 in			
R	1	4 × 4 × 10-in. plug	v	1	10$\frac{1}{2}$-in. × 5 ft. 4-in. No. 16 sieve liner
S	1	1 × 2 × 6 in			
T	2	Angle irons	w	1	9$\frac{1}{2}$-in. × 5 ft. 4$\frac{1}{2}$-in. No. 16 sieve liner
U	2	Hanger irons			
a	2	$\frac{5}{8}$ × 11-in. bolts	4a	8	$\frac{1}{2}$-in. cast washers
b	1	$\frac{5}{8}$ × 9-in. bolt	aa	10	Grates
c	1	$\frac{1}{2}$ × 11-in. bolt	bb	2	$\frac{3}{4}$-in. cast washers
d	1	$\frac{1}{2}$ × 6$\frac{1}{2}$-in. bolt	cc	4	$\frac{5}{8}$-in. cast washers
e	2	$\frac{1}{2}$ × 1$\frac{3}{4}$-in. bolts.			

grates. Notches are cut in the bottom of the partition or baffle-board, so as to permit the water to drain out from the hutch part proper when the tank is emptied of the jigging water that has become too dirty for further use.

The sieve is generally lined with No. 16 sheet iron, although sometimes this lining is as heavy as No. 12. A lever with a hook is provided for fastening to the sieve and pulling it out of the way while the hutch product is being cleaned out of the jig. The parts are proportioned so that a rougher-jig sieve filled with a bed about 8 in. thick is about balanced when the bed is submerged. A stroke of about 1 in. is used on the rougher-jigs and one of about $\frac{1}{2}$ in. on the cleaner-jigs, but the stroke is varied according to the ore and the operator.

Grates with heavy bars are used on the sieves as shown in the drawings. Slots from $\frac{5}{8}$ to $\frac{3}{4}$ in. wide are generally used on the rougher-jigs and from $\frac{1}{4}$ to $\frac{3}{8}$ in. on the cleaner-jigs. A man operating a cleaner-jig has an easy time taking care of the middlings coming from the sieve and

the box of two rougher-jigs. Usually three men will treat 15 to 20 tons of ore on two roughers and one cleaner in a 10-hr. shift, provided the ore is favorable to hand jigging. Ore suitable for hand jigging must be comparatively free from clay and the metallic minerals not too intimately associated with the gangue. The middlings are saved and re-treated in mills, while often the tailings are run over again in a mill at some later period in the history of the mine.

Ordinarily the ore, before being sent to hand jigs, has to be washed free from mud and clay, so as to facilitate the hand sorting of the coarser sizes as well as to aid in the hand jigging. This washing is done in a sluice box, in which the man wades around and claws the ore about with a shovel and his feet. After being washed, the ore is taken by the cull-man, who either screens it through openings from 1 to 1¼ in. wide, or else uses a cull fork having openings about 1⅛ in. wide between tines. With such a cull fork he removes the coarser pieces from the product that goes to the hand jigs. When the ore is coming from a "jack" mine that is later to be equipped with a mill, seldom is there enough mud and clay in the ore to make it pay to sluice the ore before hand jigging.

No bed is used on the rougher-jigs, and each time that the load in the sieve has been properly bedded, the tailings are scraped down to the middlings or "chats"; the chats are then scraped off and saved, then the bed of "jack" or lead is removed, and the sieve loaded again. Consequently, the hutch work has to be re-run to clean it, although on the sieve some clean jack or lead may be made. In case that a lead-jack ore is being jigged, the ore on the sieve is stratified into the following products; tailings, jack chats, jack, jack-lead middlings and lead. The jack and jack chats are cleaned on the cleaner-jig apart from the lead products, the two being piled separately and saved until enough have accumulated for final treatment on the cleaner. The clean lead is put into the lead concentrate and the lead-jack middlings piled separately until the time when there is a good thick bed of lead on the cleaner. The jack is then separated from the lead by stratifying it on top and scraping it off as a high-grade product to be re-run with the other jack products. The bed maintained on the cleaner-jig varies from 2 to 6 in. in thickness, as the operator lets the bed accumulate, loading on more material from the rougher and scraping off the tailings as soon as he has stratified the load properly, until finally the load on the sieve becomes so heavy that he cannot jig it properly. Then he cleans his screen, leaving a bed about 2 in. thick, and begins over again. The shake that he gives the sieve varies with the thickness of the bed on the jig. Owing to the thinness of the bed, the hutch product is not properly cleaned the first time through. Toward the end of the shift the operator of the cleaner has a thick bed on his jig and re-runs the hutch product. Of course, the hutch has to be cleaned

every time that the jigman changes from the cleaning of lead to the cleaning of zinc products. Generally, though, there is only a one-mineral separation to be made on the hand jigs.

Surprisingly clean concentrates are obtained from the hand jigs when the ore is free, as it is called when there is not much true middling going to the jigs, but inasmuch as the ore is not crushed before hand jigging, much fine mineral is locked up in the tailings, and, of course, an efficient saving is not made on the finest of the mineral particles. On account of the inclusion of some of the true middlings in the jack concentrates, these concentrates are generally somewhat lower in grade than they would be from a mill treating the same ore, but owing to the freeness with which the lead particles break from the jack and the gangue, the lead concentrates from the hand jigs are as clean as those from the mill jigs.

The hand jigs are sold ready for use, and cost completed about $30. When the ore is given a preliminary sluicing, one cullman at $2.25, one sluiceman at $2.50, and three jigmen at $2.50 each are required, making a total of $12.25 for labor per 10 hr., which may be taken as the cost of treating 18 tons of ore, so that the cost is less than 70c. per ton of ore treated.

Harz Jig Improvements.—The details of construction of the Harz jigs which were installed in the Overstrom section of the mill of the St. Louis Smelting & Refining Co. are shown in Figs. 90 and 91. The ore of the southeastern Missouri lead district is treated in this section. In this ore the lead occurs as galena in a gangue of dolomite.

There are two jigs in each section. One treats the sizes from 12 to 6 mm., the other the pulp from 6 to 3 mm. The two jigs treat about 600 tons per day. Only a gate concentrate is made, and what comes from the hutches is fine material carried over with the oversize from the trommels, together with a small quantity of fines that results from the abrasion of the bed. This hutch product is cleaned on a table. The size of the jig screens is 3×5 ft. The jigs are bulit with one compartment, but a division board is carried across the sieve to divide the jigs in two parts as far as the bottom bed is concerned, and a similar partition is put into the hutch. In this way a richer and a poorer product can be drawn from the gates, even if it is but a one-compartment jig.

The screen is supported by and nailed to the usual frame with cross slats, so as to prevent boiling of the bed. Owing to the size of the jig compartment, the sieve is given a slope of ¼ in. in 36 in. toward the draw-off side while the feed end of the screen is 1¼ in. lower than the tail end. This is accomplished by placing wedges under the screen frame and on top of the ledge or bottom-lining plank on which the screen frame rests.

The plunger is 1 in. larger than the jig screen and to eliminate any tendency to rock on its rod is secured to the eccentric by a plunger rod

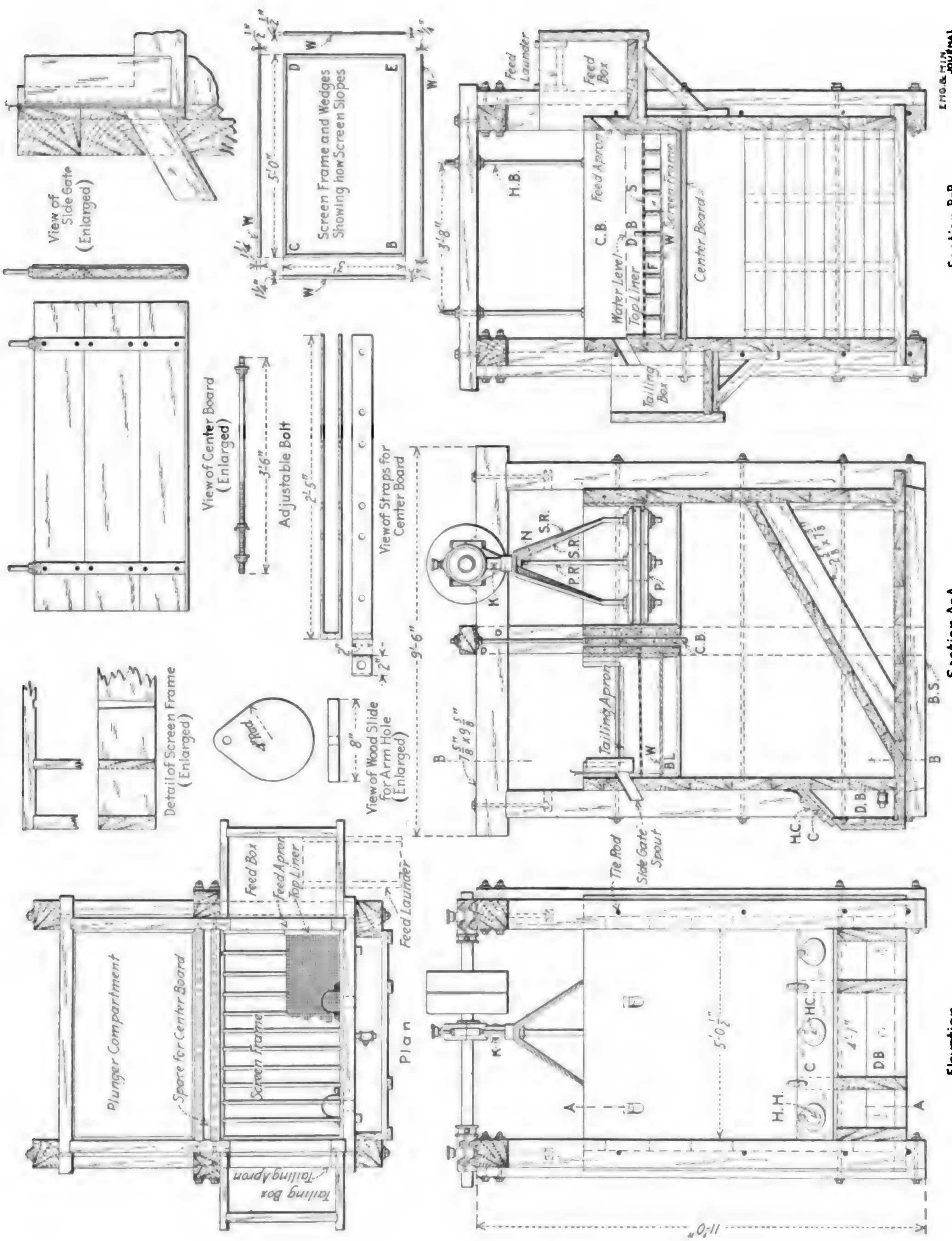
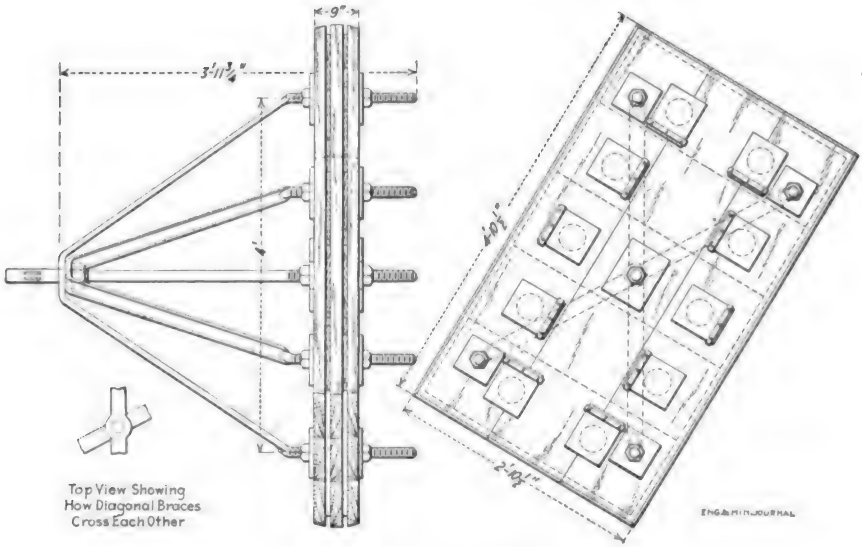


FIG. 90.—OVERSTROM MODIFICATION OF HARZ JIG.

having four strap-iron braces. The plunger rod is attached to the eccentric by a key on top and a brass nut underneath. In order to take off the plunger when making repairs, the nut is loosened and then the key is driven out. This is a somewhat more convenient way of attaching the plunger to the eccentric yoke than by using two nuts and a jamb nut. Moreover when the plunger is once set, it can be taken out and put back in the same position without binding in the corners as is apt to be the case when the plunger is secured by the other system. In the Overstrom method, owing to the driving-in type of the key no jamb nut or lock-nut



DETAILS OF JIG PLUNGER.

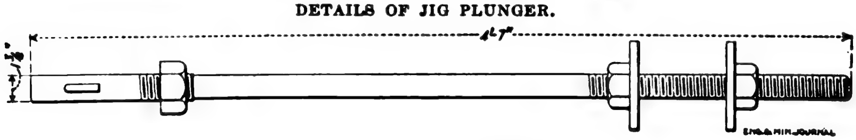


FIG. 91.—JIG PLUNGER ROD.

has to be used. The eccentric is of the slip type, but has the advantage that the heads of the set bolts face the driving shaft and so cannot catch on the clothes of the attendant.

The most interesting feature of the jig, besides this improved safety eccentric, is the movable center board in the partition between the jig and the plunger compartment. This center board is swung from two hanger bolts that pass through the top strap of the center braces of the jig frame. It is held securely in the place where it is set, as a nut is used below as well as above the carrier beam. Owing to the use of the center board, the permanent partition between the plunger and the jig compartments is

carried down only to the bottom of the bottom liner of the jig compartment, and all additional depth of partition is obtained by dropping the center board. By dropping the center board the depth of this dividing partition, upon which depends the evenness of the currents throughout the jig bed, is regulated so that the best results can be obtained and maintained. This is far better than the use of a permanent partition, which, in case a mistake is made in the original design, has to be chipped out if it extends too low and must be added to if too short, the plunger having been first taken out.

It will be observed that the binding timbers of the jig are so framed that the bottom sill does not extend out beyond the foot of the uprights, and so there is no danger of the attendants tripping. The body of the jig is made up of 4-in. planks laid on their sides. The joints are made water-tight by white lead and splines. The hutch draw-offs discharge into a box on the side of the jig. In order to prevent shavings and trash from getting into these boxes they are fitted with hinged covers in which are hand holes. These handholes are 5 in. in diameter so that they are plenty large enough to allow the attendant to put his hand down and see how the hutches are discharging. The handholes are covered by slides. This cover slopes at an angle of about 45 deg. A walkway is carried along the sides of the jigs about 6½ ft. above the floor on which the jigs stand.

The jig is tied together by 1-in. rods with the exception of the two that run through the jig under the boards of the middle partition. The position of these tie-rods is shown by the black circles in the drawing.

The plunger is built with six flap valves operating on holes in the bottom 3½ in. in diameter, so that there will be practically no suction in the jigging, for, owing to the fact that the concentrates are to come practically all from the screen and little hutch work is to be done, pulsion movement of the bed only is desired. In case suction is also wanted, the intensity can be controlled by the number of these valves left working.

Saving Wear on Hancock Jigs.—A great deal of unnecessary wear is caused and trouble experienced by having feed fall into the space between the moving frame and the tank of Hancock jigs. This is easily prevented by attaching old pieces of rubber belting on top of the tank. The belting being pliable causes practically no resistance to the movement of the frame and at the same time closes the space between the moving frame and tank to any feed material which may splash over.

Method of Fastening Screens on Hancock Jigs.—The method illustrated in Fig. 92 is used in the hard-rock mill of the Bertha Mineral Co., at Austinville, Va., to hold the Hancock jig screens firmly in position. A bearing strip is forced down on the upper battens by wedges. The battens are cambered ⅛ in. at the center, so that when the wedges are driven into position and the battens forced down upon the screen the greatest

pressure is exerted at the center where there is the greatest tendency for the screens to become loose. A wire is drawn about each pair of upper and lower battens. This method has given thorough satisfaction.

Steel Tray and Support for Hancock Jigs.—A steel tray and steel support are used for the Hancock jigs in the No. 3 mill of the Doe Run Lead Co., near Rivermine in the Flat River district of southeastern Missouri. The steel supports are shown in detail in Fig. 93, and are said to be preferable to wooden supports. The steel tray that carries the jig screens is shown in detail in Fig. 94. The screening surface of a steel tray is 4 in. longer and wider than the surface of a wooden tray. To the sides of the tray auxiliary angle irons are riveted to which the hangers are attached, for it has been found that if the hangers are attached directly to the tray and no angle irons are used, the sides of the frame may be dis-

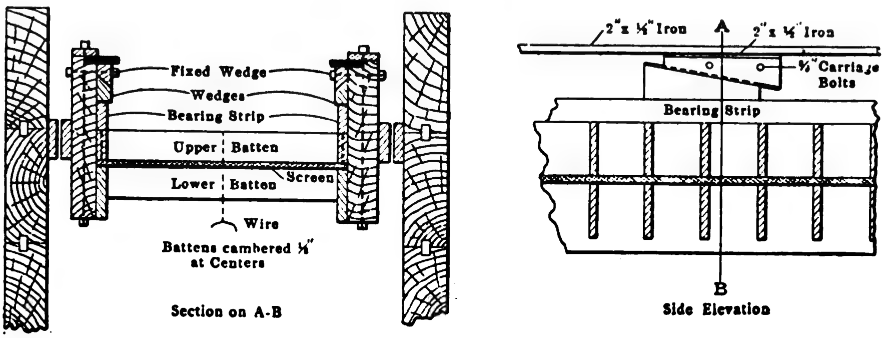


FIG. 92.—BATTENS AND WEDGING FOR JIG SCREENS.

torted when the connecting bolts are drawn tight. The auxiliary angles are separated from the upper angle iron of the frame by using spreaders of 1-in. pipe over the hanger bolts.

The steel frame of the tray is tied together by cross braces at the top, but no braces are required at the bottom because the slats that are used on the top of the screens to prevent the bed from creeping and to reinforce the screen against vibration, afford all the bracing necessary. The screens are tacked to wooden frames which are made in 5-ft. sections as shown in Fig. 94, each consisting of two end pieces of 2 × 3-in. oak. The end pieces are notched at 5-in. intervals to take the 1 × 3-in. oak cross slats, which are put in with the greater dimension vertical. These frames, are supported in the bottom of the tray by the lower longitudinal angle iron.

The top slats which rest upon the tops of the screens and prevent the bedding or "ragging" from creeping are made of oak in sections of two slats as shown in Fig. 94, the greater dimension of which is vertical as is the case with the lower slats. The end pieces of each section extend out

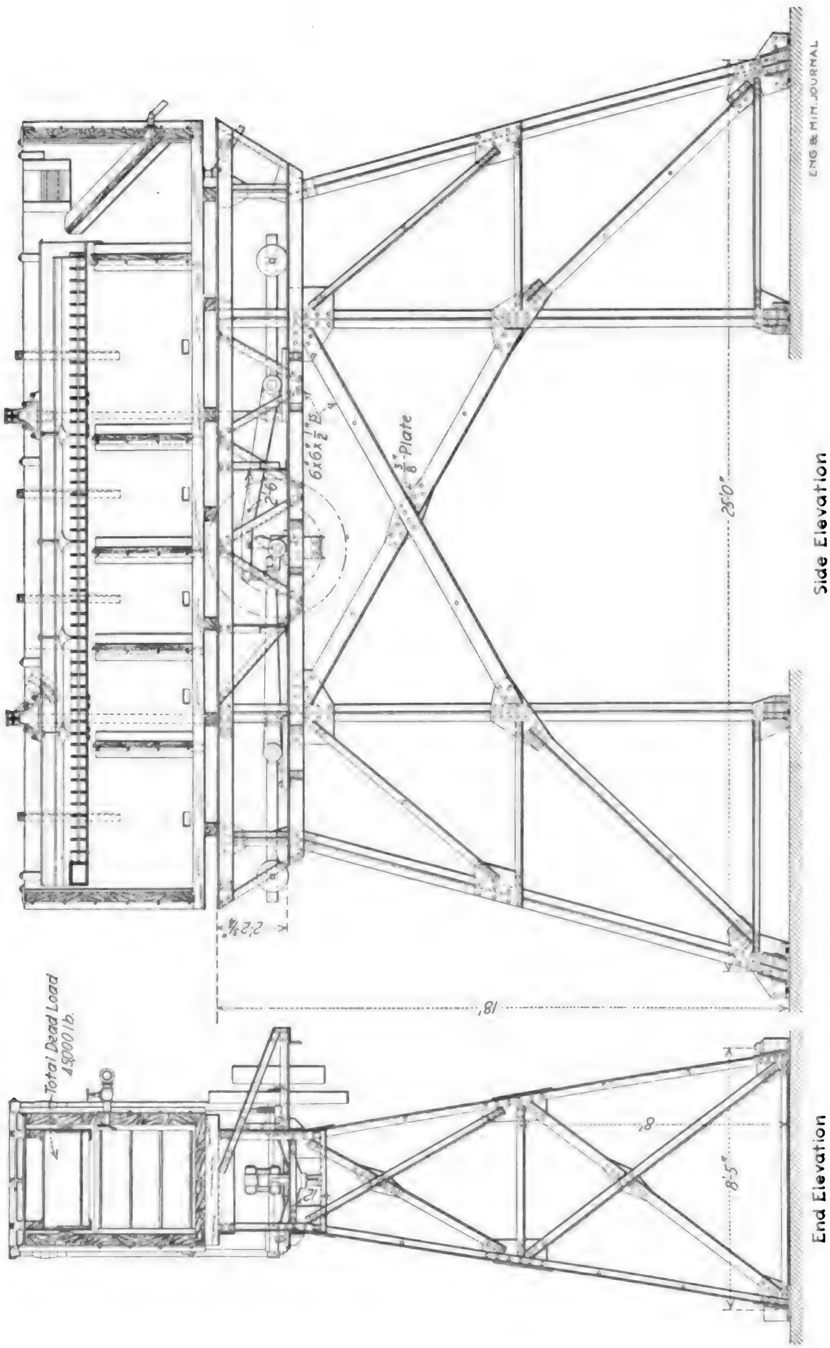
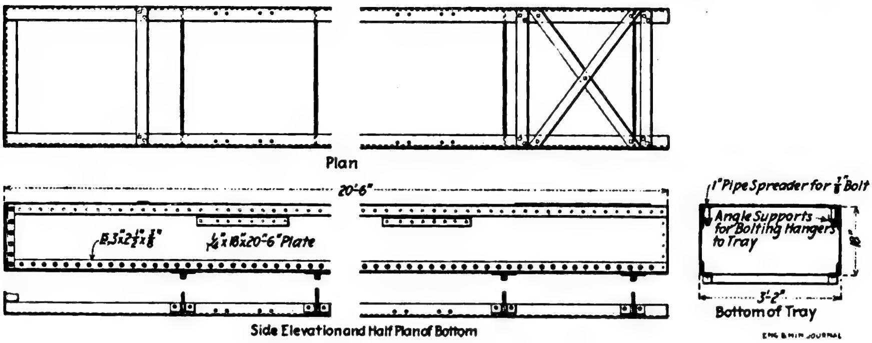


FIG. 93.—HANCOCK JIG AND STEEL SUPPORT USED IN DOE RUN NO. 3 MILL.

beyond the sides of the slats for 2 in. or half the width of the opening between slats. The slats are protected from wear from the rubbing of the ragging and the movement of the pulp by a piece of iron plate $\frac{1}{8}$ in. thick that is screwed to the back side of the slat and by a $1 \times 1 \times \frac{1}{8}$-in. angle iron on the top and front side. The sections of slats are held in place by key-boards or pieces of 2-in. pine 8 in. high that are placed on top of the slats



STEEL TRAY FOR HANCOCK JIGS.

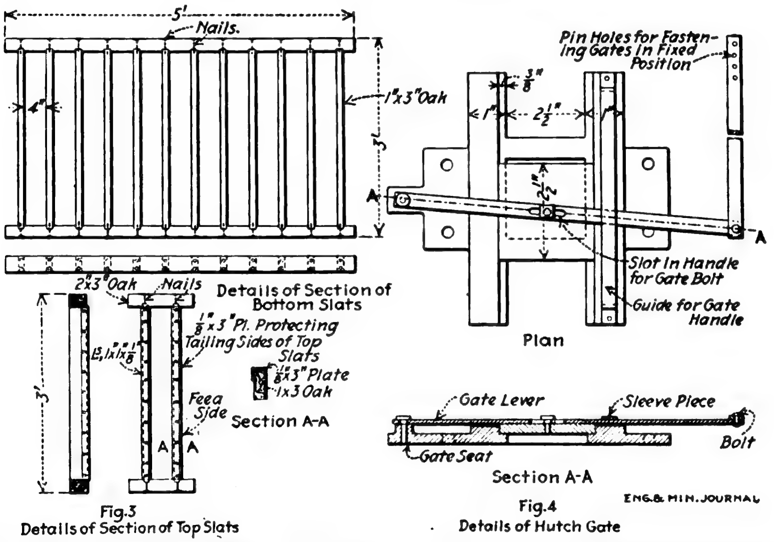


FIG. 94.—SLATS AND HUTCH GATE USED IN HANCOCK JIGS OF THE DOE RUN MILL.

at the ends and parallel with the long side of the tray. The keyboards are wedged into place by a series of double wedges about 10 in. long sawed from 2-in. pine boards; the wedges being used between the top of the key-board and the bottom of the upper angle of the tray. This system of keying facilitates removal of the upper slats when screens have to be changed.

The type of gate used to control the discharge of middlings and con-

centrates from the hutches of the jigs used in the Doe Run and St. Joseph mills is also shown in Fig. 94. These jigs are operated with constant discharge of middlings through the partly opened hutch gate.

The Hancock jig is usually supplied with clean feed water that enters at the hutches and flows upward thus introducing the classifying power of a stream of water as a factor in jigging. This upward flowing stream has the effect of partly offsetting or neutralizing the strength of the suction. The supply of feed water is automatically controlled at the Leadwood mill of the St. Joseph Lead Co., by a float in the tailing discharge compartment which acts upon a butterfly valve in the feed-water pipe.

The Doubledee Plunger (By Lucius L. Wittich).—It is reported that the Doubledee hopper-shaped plunger, adapted to any make of jig

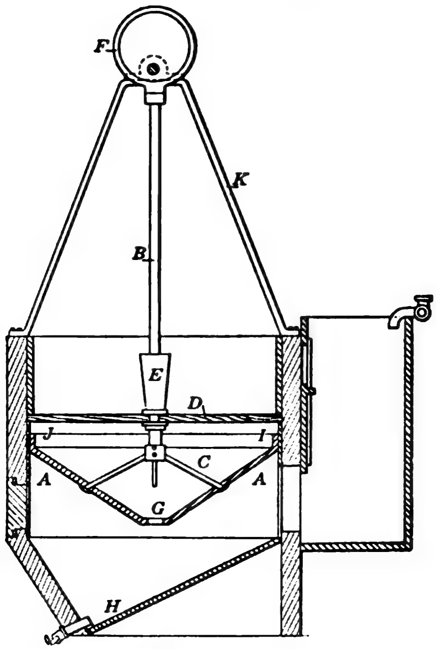


FIG. 95.—DOUBLEDEE JIG PLUNGER.

for treating zinc ores, has proved a success at the mill of the Little Anna Mining Co. on land belonging to the city of Joplin, Mo. So radically different is the Doubledee attachment from the ordinary plunger that its work is being watched with keen interest throughout the Joplin zinc and lead district. The device has only recently been patented by M. Doubledee.

A sectional view of the Doubledee jig tank is shown in Fig. 95. In all other jigs in use locally, the plungers are in tanks at one side of the cells and it is the downward stroke that forces the water in the cells upward

through the sieve. This plunger *A* brings the water up with its upward motion. It is claimed that this keeps the ore constantly in motion and that the separation is made more thorough.

The plunger is made of ¼-in. boiler plate and is attached to the plunger rod *B* by spider braces *C*, made of 1½-in. round iron. The plunger rod, 2½ in. in diameter, passes up through a hole in the sieve *D* and is shielded by a cast-iron guide *E*, which is funnel shaped and sufficiently large to permit free action of the rod. This rod attaches to the eccentric *F*, which permits any regulation of the stroke. The eccentric shaft is braced by cast-iron rods *K*. For heavy loads on the roughing sieves the length of the stroke varies from ¼ to 1½ in.; for light loads on the cleaning sieves the stroke varies from 1/16 to ½ in.; while for extremely heavy loads it goes to ¾ in. The speed of the stroke also varies. The length of the stroke is much shorter than required on the average jig with ordinary plungers.

In the bottom of the hopper-shaped plunger is an opening *G*, 2 in. in diameter, through which the concentrates and fines pass into the hutch *H*. It is claimed for the invention that a bed on the sieve need be only 1¼ in. thick to insure pure concentrates. As material passing over the cells has a tendency to lump at the upper ends *I*, the plunger is set as close as possible to the wall of the cell, while at the lower end *J* a space from ½ to ⅜ in. is allowed.

Device to Reduce Top Water on Jigs (By James L. Bruce).—In the Joplin district the crushed unsized ore is partially concentrated on five- or six-cell rougher jigs and this concentrate is then cleaned on a six- or seven-cell cleaner jig. In this practice the water added to the plunger compartment of each cell increases the top water of the following cell and toward the tail end of the jig there is a race of top water. This, in addition to carrying away large quantities of ore without giving it time to settle, disturbs the pulsion of the plunger water and interferes with the proper settling of the concentrate and the bedding of the jig cell. At the Continental Zinc Co. plant the jigging action has been much improved by dewatering the lower cells of the jig with a simple, inexpensive arrangement. In principle it is an adjustable slicer which removes the top layer of water as it goes over the partition between the cells of the jig and allows the gangue and remaining ore to run under it to the next cell. Only the finest slimes are carried off with the water, and this is conveyed to settling tanks, whence it goes to the tables. The cell ahead of the dewatering device as well as those following are benefited, those following by the decrease in "top water" while the backwater on the cell ahead reduces the surface currents and provides a steadier discharge which disturbs the bed less. The top water is removed through a hole cut in the side of the jig at the end of the partition between cells and from there carried to the settling tanks which feed the concentrating tables.

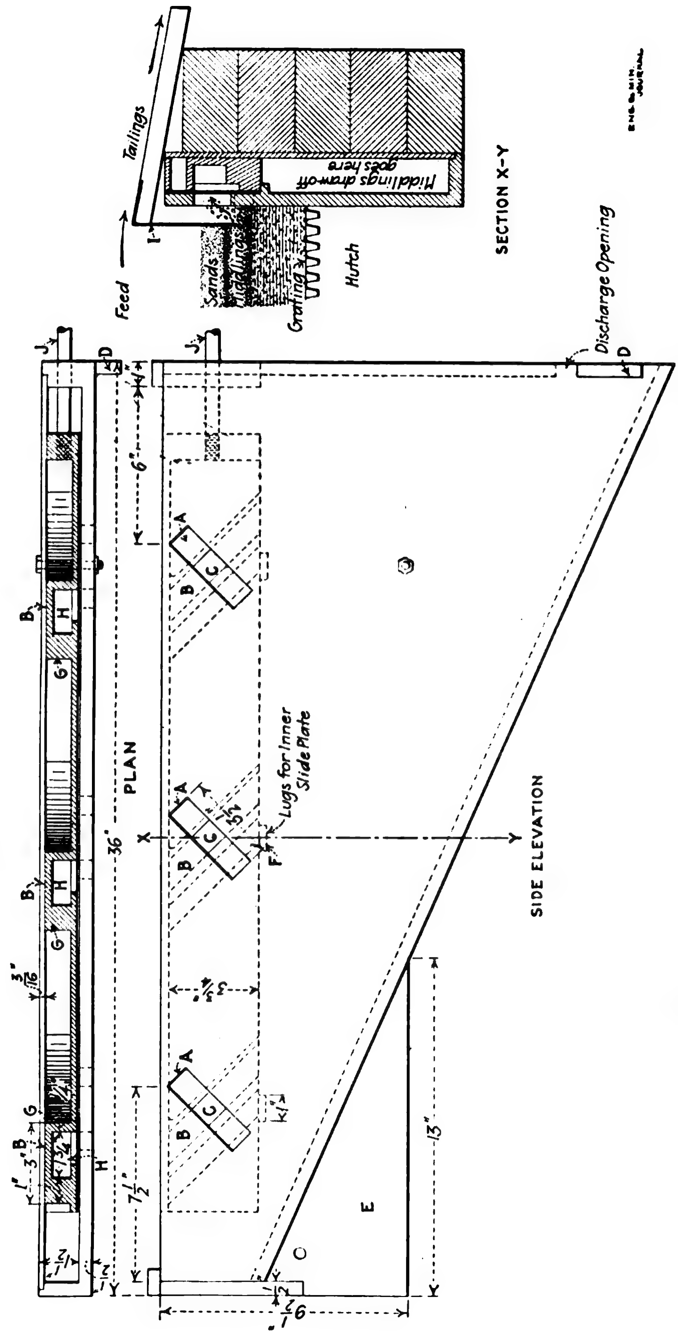


FIG. 96.—MIDDLING DRAW-OFF FOR JIG WITH ADJUSTABLE HEIGHT OF OPENING.

A cross-section of this device viewed from the plunger side of the jig is shown in Fig. 97. There are seven parts: *A* is the dam which holds the water back, causing it to flow through the opening *Z* and is made of a piece of pine $\frac{5}{8} \times 6$ in., with a length equal to the width of the cell; *B* is the rigid part of the slicer made of No. 10 or No. 12 sheet steel about 2 in. wide, fastened with screws to the under side of *A* and of the same length; *C* is the adjustable part of the slicer and can be set to remove as much or as little of the top water as desired. It is made of No. 10 or No. 12 sheet steel, about 4 in. wide, and the same length as *A* and *B*, with two or three lugs, which project from one side and through slots in *B* for hinges;

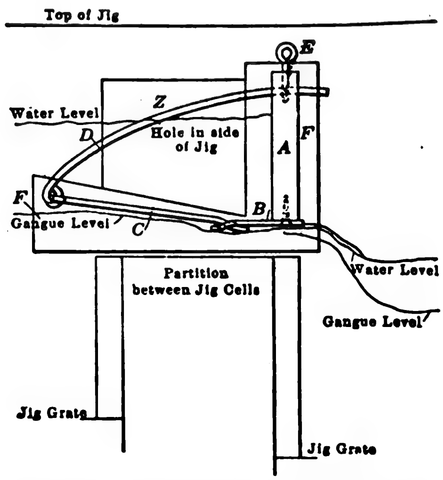


FIG. 97.—DEWATERING DEVICE FOR JIGS.

D is a heavy wire for adjusting *C* and passing through a hole near the top of *A*. It is held in place by a nail or pin *E* in a hole alongside the hole through which *D* passes; *F* is one of the two end pieces which fasten the device to the sides of the cell and is cut with a bevel to keep any water from going out through the discharge *Z* below the slicer *C*.

Adjustable Draw-off for Jig Middlings.—The devices commonly used to draw off the middlings from a jig bed have the height of the opening fixed. A draw-off patented by C. E. Knowles and used in some of the Joplin mills permits the opening through which the middlings pass to be raised or lowered. It consists of a fixed casting set across the lower end of the jig compartment with three slots *A*, referring to Fig. 96, cut at 45 deg. in the upright face and of another casting set inside the fixed casting and capable of movement across it, with three similar slots *B* inclined in the opposite direction. The faces carrying the slots are in contact and where the slots cross, a diamond-shaped opening *C* is formed, through which the

concentrates pass. As the movable casting slides back and forth, the opening is raised and lowered. The fixed casting is in the form of a box, a cover plate forming the vertical side which does not carry the slots. The dimensions and shape are as shown. The middlings fall to the sloping bottom and are discharged at the lower end. A lug *D* and a bolting bracket *E* are provided for holding the box in the jig compartment. Lugs *F* are cast on the face carrying the slots in order to support the sliding casting. The sliding casting is made with inclined lugs *G* on the back

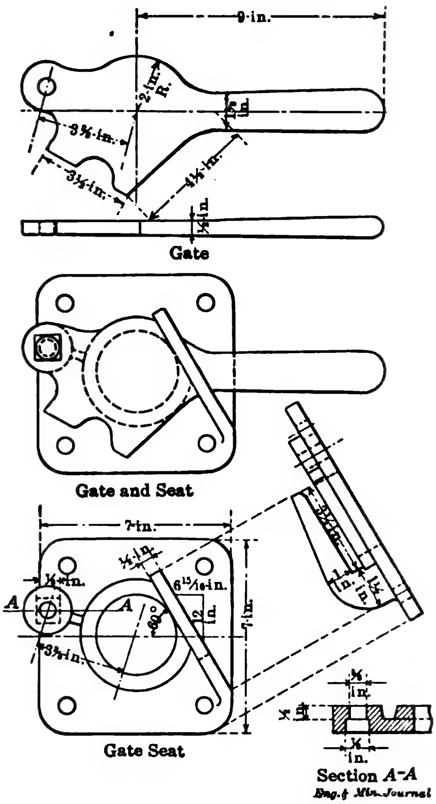


FIG. 98.—TAILING GATE USED ON JOPLIN JIGS.

which follow down so as to correspond to the slots in the face and bear against the cover plate of the fixed casting. They are hollow, connected with the slots and are of a size, so that the chutes *H* which they form are somewhat larger than the slots. The middlings passing through the diamond-shaped holes slide down these small chutes and fall to the sloping bottom of the fixed casting. The top of the sliding piece is a plate to fit the inside of the fixed casting. The movement of the piece is controlled by the rod *J*, which is screwed into its end and projects through the side

of the jig. In setting this discharge device in the jig compartment, a hood *I* is brought down from the tail-board discharge between the discharge device and the jig bed to such a point that the middlings only can pass under it and be drawn off. This is illustrated in the cross-section of a compartment end.

Tailing Gate for Jigs.—In the Joplin district, whether a simple settling box or one fitted with a hydraulic attachment is used, as a settling compartment at the tail end of jigs, these compartments are fitted with gates of the general type shown in the Fig. 98. The gate consists of a cast-iron plate that is bolted to the wooden frame of the jig. In it is a hole 3 in. in diameter, while projecting from its side is a brace to support the gate proper which is fastened to the seat casting by a bolt. The gate piece is cast with a handle to permit of easy adjustment, while in the bottom face is a small semicircular recess. This permits the opening to be contracted as much as desired, and it is possible, in case of stoppage, while using a small discharge opening, to start the flow again by simply opening the gate valve to its full extent. It is by the size of the draw-off opening that the grade of the overflow is regulated, and it is on that account that the valve must permit close adjustment, while, because of the fact that the bulk of the mill feed goes through it, it must be cheap and easily replaceable. The gate herein described seems to fill these requirements.

Device for Clearing Jig Grates.—The chats made in jigging ores from the Joplin district because of their angular shape clog jig screens to an annoying degree. Therefore in most of the mills, cast-iron or wooden grates are used in lieu of screens to support the bed of material in the jigs. The cast-iron grates are made as long as the jig compartment is wide, and about 6 in. in width. But even the grates become clogged, making it necessary to dig down into the beds to dislodge the caught particles about once every hour. This is often done by inserting a blunt chisel-like tool about 2 in. wide through the bed and into the slots of the grates. This operation is known as spudding. As the mill attendants are inclined to neglect this work, S. C. Byrd and C. Pierce have devised and patented an automatic spudder. As shown in Fig. 99, the device is held in place by two 60-d. nails that pass through holes in the shoe and extend down into the open spaces between the bars of the grate. A 1-in. pipe *F*, 18 in. long, fits snugly over the upper extension of the shoe *E*, forming a cylindrical chamber in which a $\frac{3}{4}$-in. rod *D* moves up and down. An eccentric on the jig shaft moves the horizontal arm *A*, which engages the rocker *B*, mounted on a shaft *G*, that is supported by bearings mounted on the partitions between the cells of the jigs in such a manner as not to interfere with the flow of pulp. The rocker *B* actuates the rod *D* by the arm *C*, which is fixed to the rocker shaft *G*. When the eccentric is approaching its position of maximum forward throw, the

shoulder on the arm *H* engages the rocker *B*, causing it to move outward and to lift the arm *C* and the rod *D*. As the eccentric turns past that position, the shoulder on the arm *A* is withdrawn from the rocker *B*, so that it returns suddenly to its normal upright position, permitting rod *D* to drop freely upon the grate at the time that the plunger of the jig is making its downward or pulsion stroke. The jar of the rod dropping on the grate loosens the chats caught in the slot, so that the upward flowing current of water will carry them up into the bed. The throw of the eccentric is about 1 in. A spring is attached to the arm *A*, as shown in the illustration, to prevent chattering of the arm in the slot of the rocker. So

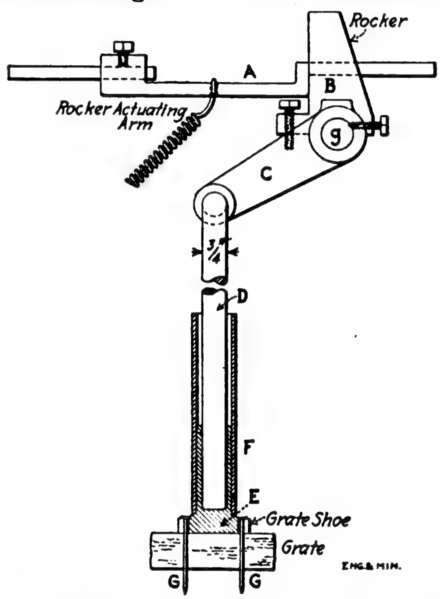


FIG. 99.—DETAILS OF GRATE-CLEANING MECHANISM.

far the device has been used only on the roughing jigs, no attempts having been made to use it on the cleaning jigs that are operated at greater speed. The area of each compartment of these jigs is 36 × 48 in., and eight grates are required for each compartment. One arm, *C*, one rod *D* and its shoe *E* are required for each grate, but only one eccentric, one arm *A* and one rocker *B* for each compartment.

A New Jig Grate.—A new jig grate designed by Fred Richardson is being experimented with in the J. L. Sullivan and Little Princess mills at Oronogo, Joplin district, Mo. The grate is made of cast iron and it differs from the regular cast-iron grate in the shape of the bars. As shown in Fig. 100, the bars are ½ in. wide, 1⅛ in. deep and spaced about ⅛ in. apart. The object in making the bars with a bevel or slope on one side is to give a backward, as well as upward motion to the water.

This tends to retard the progress of the heavy mineral on the screen. By retaining the material on the bed a little longer, less of the middlings pass from one compartment to the next. The backward current of the water, as shown by the arrowmark *c*, also reduces the friction at the lower end of the screen so that the mineral has less tendency to work over

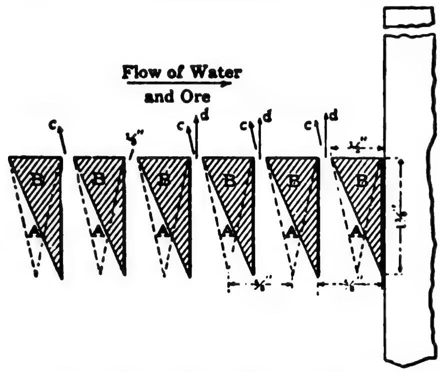


FIG. 100.—RICHARDSON JIG GRATE.

the bridge. In using this grate about 3 in. of ore are kept on the screen. Of this 2 in. are concentrates and middlings which form the bed. The chats occupy about 1 in. above this, while there are 2 or 3 in. of water. The feed and water used are practically the same as with the regular

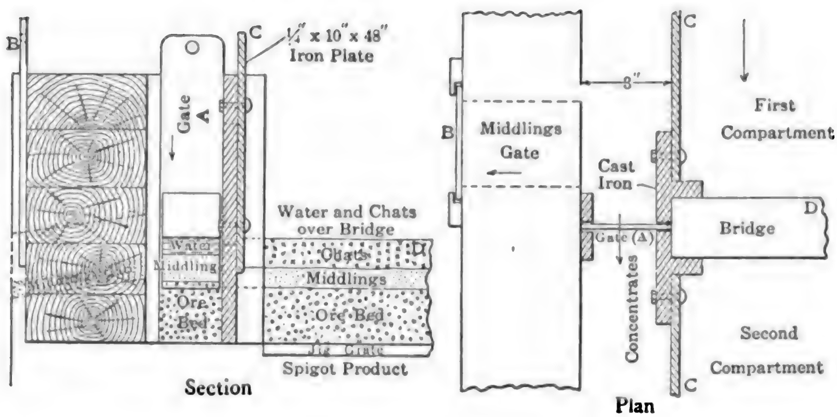


FIG. 101.—RICHARDSON JIG GATE.

grate. The new grate does not increase the capacity of the jig but it is claimed to give a cleaner extraction of the ore. One analysis of the tailings showed that they contained only 0.34 % zinc. This is much cleaner than the average concentrate produced in the Joplin district.

Another feature that is being worked in connection with this grate is a gate or bypass whereby a portion of the heavy material from one compartment passes to the bed of the one following without going over the bridges. This is accomplished by means of a gate occupying about 3 in. at one end of the bridge. Along one side of the jig cell is a $\frac{1}{4}$-in. steel plate *c*, Fig. 101, about 10 in. wide. The steel plate is adjustable and can be placed at any depth in the bed, usually $1\frac{1}{2}$ to 2 in. from the grate. This allows the mineral particles to pass under the gate and separate them from the chats. A portion of the middlings is drawn from the cell by the usual side gate *d*. The main advantage that seems to arise from this bypass is that middlings and concentrates may pass from one cell to the next without passing over the bridge. In other words, it maintains the concentrates and middlings at a lower level and holds this portion on the grate longer and at the same time allows the chats to pass off in the usual

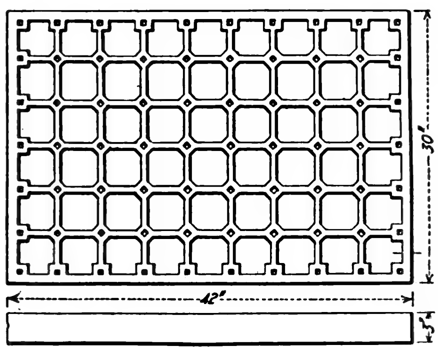


FIG. 102.—JIG GATES USED IN JOPLIN MILLS.

way. In this particular case the concentrates are not drawn off until at the last cell. There should be an arrangement to take at least a portion out at an earlier stage for there is a more or less abrasive effect on the ore and this will result in the loss of mineral.

Cast-iron Screen Frames for Jigs (By Claude T. Rice).—Grates are generally used to support the bed of ore in the jigs in Joplin mills. They are preferred because the cherty constituents of the ore tend to break into long splinters which choke wire-cloth screens; the grates do not become choked as easily as do screens and when choked can be more easily cleared. However, in certain mills screens are used in some of the jigs, and in order that the jigs may do their work most effectively it is essential that the screen be supported rigidly in a horizontal plane. In many jigs a wooden grating is used for this purpose, the screens being stretched tightly over and tacked to the grate.

Another device for supporting wirecloth screens has been developed by J. A. Rogers, a Joplin foundryman. It consists of a rectangular cast-iron grid, 1 in. wide by 3 in. deep, divided by ribs into a number of lozenge-shaped openings. The ribs are $\frac{5}{8}$ in. wide and 3 in. deep. The junctions of the ribs with one another and with the sides of the rectangular frame are enlarged and in casting the grid these enlarged junctions are cored so as to leave holes $\frac{1}{2}$ in. square in the finished casting. At the junctions of the ribs with the frame the sides of these squares are parallel to the sides of the frame while at the junctions of the ribs with one another the diagonals of the square holes are parallel to the sides of the frame. Woodenplugs are driven into these small holes and after the projecting ends have been cut off flush with the top of the ribs the screen is tacked to these plugs. Fig. 102 illustrates the form of the screen frame.

Overstrom Jig Eccentrics.—The details of the jig eccentric designed by G. A. Overstrom, are shown in Fig. 103. These eccentrics are used

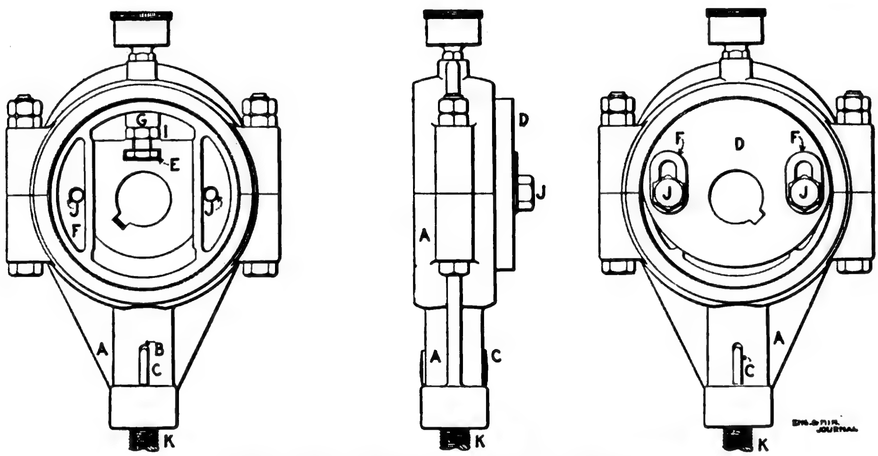


FIG. 103.—THE OVERSTROM SAFETY JIG ECCENTRIC.

on the large single-compartment, roughing or bull jigs in the Overstrom section of the mill of the St. Louis Smelting & Refining Co. in southeastern Missouri. The screens of these jigs are 3 ft. wide by 5 ft. long. The special feature of these eccentrics is that the adjusting nut for changing the throw is in the interior of the eccentric and does not stick out where it might strike an attendant while oiling or doing other work about the top of the jig. The cap screws on this eccentric project a little from the hub, but as they are turned toward the driving pulley of the shaft, are not in a position where, under ordinary circumstances, they could catch upon the garment of an attendant. The strap yoke contact with the eccentric was made flat simply because of the tools that were at hand. Where the machinery is available for making ball-and-socket

surfaces, Mr. Overstrom makes the joint between the eccentric and the strap of that type, as there is no possibility of the plunger binding on the sides of the plunger compartment of the jig.

The eccentric yoke is attached to the plunger piston by the key *C*, that passes through the slot *B*. The hub *G*, that passes through the eccentric is made ovoid. This hub, of course, is keyed to the jig shaft. Two slots *F*, are cut through the collar of the hub where the hub proper goes inside the body *K*, of the eccentric and is fastened to it by means of the two cap screws that pass through the slots *F*. In the hub *G*, is a slot *D*, cut out to receive the adjusting nut *I*, it being recessed to grasp the head of the adjusting nut, so that the hub is held securely in position when the clamping nut is brought to bear on the upper face of the hub. The body of the eccentric is cut out in the center so that it will straddle the hub portion of the eccentric. This hub is inserted in the body, the cap screws *J*, are screwed into the body after being inserted in the slot *F* and then the adjusting screw *I* by the nut on it is screwed out so as to enter the boss *G*, on the body of the eccentric. The adjusting screw *I*, is screwed into the body until the desired eccentricity is obtained. Then the locking nut is screwed home on the hub and the body locked securely into position with respect to the hub. Next the cap screws are tightened to grip the hub and the body together, and the eccentric is set rigidly.

FINE SAND AND SLIME CONCENTRATORS

Vanners and Bumping and Jerking Tables

Reclaiming Zinc and Lead Slimes (By Lucius L. Wittich).—The necessity of reclaiming the greater percentage of the economic minerals, if the thin sheet-ground mines in the Missouri-Kansas-Oklahoma district are to be operated at a profit, has caused the larger companies of the district to install "sludge" departments for the treatment of slimes; but the operators of the mines of the American Zinc, Lead & Smelting Co., at Webb City, Mo., have gone one step farther and are experimenting with the treatment of overflow waters from the sludge tanks. An additional plant, equipped with tables made by the James Ore Concentrator Co., of Newark, N. J., has been constructed, and at a small expense for operation, the new plant is producing on an average of 3000 lb. of galena and 6000 lb. of blende concentrates each week.

In former years the waste product from the first milling went to the dump heap and the water, carrying in suspension a heavy volume of zinc and lead, went into the sludge pond. As a result it has been found profitable, in many instances, to re-treat the tailing piles, to clean out sludge ponds and remill the sediment and even to grind up the old boulder heaps that were once considered worthless; but the present experimental

plant of the American company is the first undertaking to effect higher recovery by treating on concentrating tables the mill water, already passed through a series of settling tanks and apparently having had all the sulphides extracted. After leaving the mill the water goes through three settling tanks, from which the sediment is treated on a series of tables in the first sludge mill. The overflow from these tanks then goes to a series of 10 new tanks, and although it is apparently free from metallic sulphides when leaving the first set of three tanks, the additional settling process has proved successful, and a recovery of both zinc and lead has been made, as mentioned above.

A Vanner Regulator (By John Tyssowski).—Millmen on the Mother Lode of California have evolved many labor-saving devices for the regu-

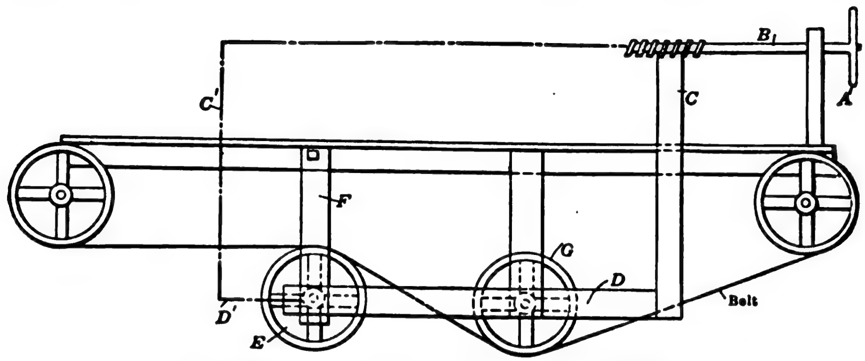


FIG. 104.—REGULATING DEVICE FOR VANNERS.

lation of mill machinery. The device for adjusting a vanner belt, shown in Fig. 104, is used in the 60-stamp mill at Melones, near Angels Camp, Calif. On the ordinary vanner concentrator the belt travel must be regulated by handscrews on the side of the table. This makes it inconvenient for the vanner man to perfect the necessary adjustment and takes up a lot of unnecessary time in climbing between the tables. With the arrangement shown, the travel of the belt is regulated from the end of the table by a wheel that is operated from the aisle between the tables. To adjust the travel of the belt of the vanner it is only necessary to revolve the wheel *A* which is connected by the rod *B* to the wooden rod *C*, screwing into the latter so that by rotating the wheel *C* is swung in either direction. The rod *C* is pivoted on the frame of the table and communicates its motion to a second rod *D* which is connected to the axle *E* of the tightening roller under the vanner belt; *D* has no connection to the dipping roller *G*. The axle *E* is suspended from the frame of the table by the hanger *F*. By using a worm gear, a delicate adjustment of the table is made possible and there is much more likelihood of the vanner man keeping his tables in adjustment if there is a minimum of exertion

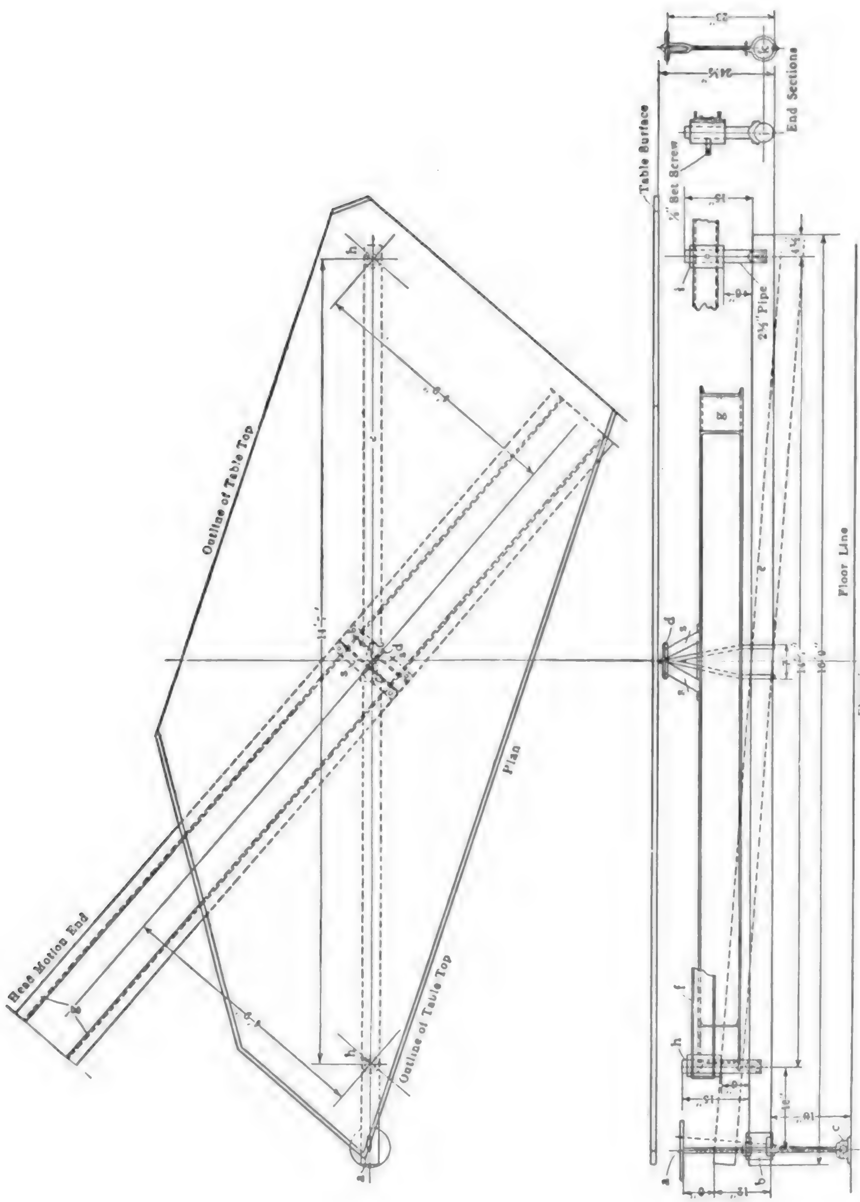


FIG. 105.—TIPPING DEVICE FOR OVERSTROM CONCENTRATING TABLE.

required for perfecting these adjustments. The dotted lines in the drawing indicate another arrangement by which the same result is obtained. The rod B is fastened to C', which is connected by a short lever D' that transmits the motion to the axis of the roller E, as in the other case. The former arrangement has the advantage of requiring a shorter iron rod, but either is satisfactory.

Rearrangement of Tipping Device for Overstrom Tables (By John Tyssowski).—So much force was required to operate the wedges provided for regulating the tipping motion on the Overstrom concentrators installed in the hard-rock mill of the Bertha Mineral Co. at Austinville, Va., that a delicate adjustment of the tables could not be obtained easily. They were therefore rendered unsuitable for the close work required. To overcome this difficulty, W. O. Borcherdt, assistant superintendent, devised the regulating device, shown in detail in Fig. 105 by which any desired adjustment of the table may be procured by simply spinning a hand wheel which communicates its motion to a 4-in. pipe hung from the frame of the table.

This hand wheel, shown at *a*, is attached to a feed screw and nut from an old 3¼-in. Ingersoll drill. This is provided with a ball joint *c* in the floor socket to allow the necessary play. At *b* there is a short section of 5-in. pipe with slot to retain the nut sleeve and permit slight lengthwise motion of the 4-in. wrought-iron pipe *e*, which is also slotted so that forward, backward and sidewise motion is permitted. The dotted lines show the extreme position for the 4-in. pipe, which has a 6-in. range of motion. Standards of 2½-in. pipe, shown at *h*, connect the 4-in. pipe to a 5-in. channel iron *f*, which partially supports the frame of the table. The 9-in. channels *g* carry the weight of the table to concrete foundations at either side, and support the standards *s*, on which the 1-in. turning pin *d* bears; the 4-in. pipe hangs from this pin by 1½-in. iron straps clamped about it with ⅜-in. bolts. This arrangement is shown at *k*. The old wedge socket of the original device is shown at *i*; *j* is a section of the 4-in. pipe showing 2½-in. pipe standard and method of securing it to the 5-in. channel. Besides making the most delicate adjustment of the table possible and easy, all but two of the concrete supports formerly required are done away with by this arrangement and much clearance is added under the table.

Protecting Riffles on Wilfleys.—At the Steptoe concentrator, Ely, Nev., George Waddell has effected a considerable saving in the cost of keeping the surfaces of the Wilfley tables in good condition by nailing protecting strips of sheet copper on top of the riffles; the copper used is 0.025 in. thick. To cover a table 8.75 lb. of copper are required, so that the copper for a table costs about $2 while the wooden strips to cover a table cost $7 per set. It takes two carpenters a day to cover a table with

linoleum, put on riffles and nail on the copper strips, while they can re-riffle two tables in a day. The riffles are nailed 8 to 10 in. apart while the copper strips, which are punched so they will not get bent during the nailing, are fastened to the riffles every 1½ in. with copper nails. In order to allow for the extra thickness of copper, the riffles are shaved down on the thin end. By means of these strips the life of the riffles has been increased four times what it was without the copper strips. The wood of the riffles eventually cuts out with use and as soon as the nails are exposed new riffles have to be put on. These strips are used only on the tables treating the coarser sizes of pulp as it is only these riffles that wear rapidly.

Wilfley Table Kinks (By Claude T. Rice).—In the mills of the Joplin district, Missouri, it is customary to use a drip spout on the Wilfley tables, on which the finest material is concentrated, to keep water flowing down the tail edge of the table. On the tables on which the coarser material is concentrated a cleat is nailed to the table near the edge of the tail

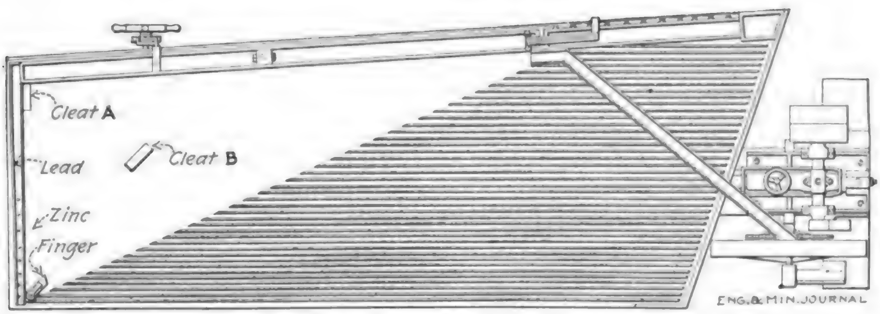


FIG. 106.—CLEATS AND FINGER ON WILFLEY TABLES.

end and under the wash-water box, as shown at *A*, Fig. 106. The motion of the table causes the water to bank up at the cleat and flow down the tail edge, where ordinarily little water flows. Should the water tend to flow away from this edge, say at a distance of half the width of the table from the corner of the wash-water box, a second cleat *B* is nailed to the table which tends to deflect the water back to the edge. This cleat is nailed to the linoleum above the space occupied by the concentrates and far enough from that space so that the water will have spread out evenly below it before the band of concentrates is reached.

In order to keep the line of separation between blende and middlings from coming to the corner of the table, where, in order to keep the concentrate clean, closer watching is necessary, it is the custom at many mills to use the finger shown in Fig. 1, illustration 107. This is bolted to the deck of the table so that it can be rotated to cause the concentrates to discharge always at the front edge of the table. This is done because if the

band of concentrates works past the corner, the line of separation between concentrates and middlings moves back and forth in relation to the dividing partition of the catch-box a distance equal to the stroke of the table. Some of these fingers are made of cast iron, others of wood with old rubber belting attached to the under side to rub upon the linoleum surface of the deck.

As the tables are used to make a three-mineral separation it is customary to provide the concentrates box with catch spouts that are supported by rods held at the tail end of the table by 2-in. pieces of wood. These sliding spouts are illustrated in Fig. 2. They are required because the line of concentrates frequently changes and with every change it is

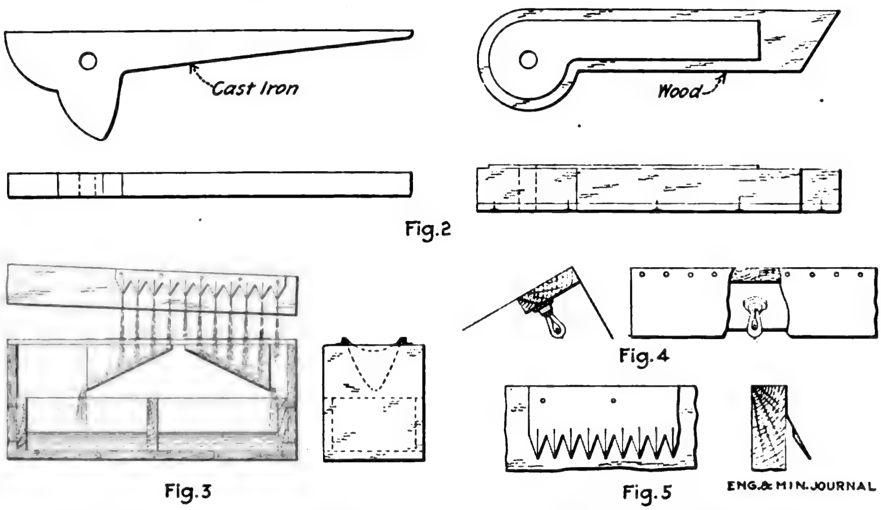


FIG. 107.—CONCENTRATES COLLECTING DEVICES AND LIGHTING TROUGH FOR WILFLEY TABLE.

necessary to adjust the spouts so that the same character of product will always be discharged into the same catch box.

The frequent changing of the line of concentrates necessitates strong lighting of the tables at night. Therefore, a lighting trough, illustrated in Fig. 3, is hung over the tail end of the table. In the trough are several 8-c.p. electric, incandescent lamps, the sockets of which are supported upon a 2 × 6-in. timber which also carries two pieces of galvanized iron 9 in. wide to reflect the light downward. The lighting troughs are suspended from the mill rafters.

In making a three-mineral separation it is necessary to make a sharp distinction between the bands of concentrates and middlings. If the concentrates come from the table in broad bands and the feed has not been closely classified, it is difficult to make a sharp separation between

two products by using only the catch spouts. To facilitate making the separation it is the practice in certain Joplin mills to nail a notched lip or piece of galvanized iron to the tail end of the table. This lip is made as illustrated in Fig. 4, the depressions down the center of each of the Vs being made by bending the iron over a 20-d. nail. A portion of the bands of concentrates equal in width to the width of the upper end of the V is discharged from the lower end of each depression, the discharge coming off the table in several drips or streams that permit making a close differentiation between products.

Nevada Consolidated Canvas Tables.—Double-deck belt canvas slime tables are being used in the Steptoe Valley concentrator of the Nevada Consolidated company at McGill, Nev. The speed of the belt of this machine is approximately 7 ft. per min. and in the double-deck type of table, the low-grade concentrates of the upper deck are passed to the lower deck for re-treatment. It is the intention of the management to work these slimers in decks of three; to take the low-grade concentrates from each deck and re-treat it on a finishing machine. It is hoped thereby to get an insoluble in the concentrates of 50% as against the present average total insoluble of 65%. The machines are about 8 ft. wide and 13 ft. long and have a capacity of about 8 tons of solids per belt per machine per day. The feed to the tables assays approximately 1% copper and carries 18% in solids and the material is all finer than 200 mesh. The feed is material all of which has been discarded by the mill. The product of the machines treating daily 80 tons of solids containing 1% copper, has been amounting to approximately 14,000 lb. of copper per month, the concentrates having the following analysis: Gold, 0.073 oz.; silver, 0.21 oz.; copper, 5.7%; silica, 5.69%; iron, 9.2%; lime, 0.8%; alumina, 9.1%; and sulphur, 8.3%. The extraction is approximately 30% on a table of this type. The material is used exclusively in claying up the reverberatories and as a smelter and converter flux and if, with some changes anticipated in cleaning the rough product, a reasonable extraction can be obtained with an insoluble of 50% or less, this material can go with the other concentrates through the roasting department.

Canvas Tables of the Combination Mill.—In Figs. 108 and 109 are shown the details and general arrangement of the canvas table, designed by A. G. Kirby, and used at the old Combination mill, at Goldfield, Nev. The notable features of this table are the distributing trough at the head end and the manner of deflecting the concentrate into a collecting launder at the end of the table when the supply of feed is shut off and the concentrate is to be gathered. The manner of collecting the concentrate and distributing the feed, is clearly shown in the drawings. The canvas tables were arranged in the Combination mill, as is shown in the sketches. The tailings from the lower end of the first table were sent to the feed

trough of a second set of tables placed below and extending out from the tail end of the first series. Later the tailings from the first set of tables

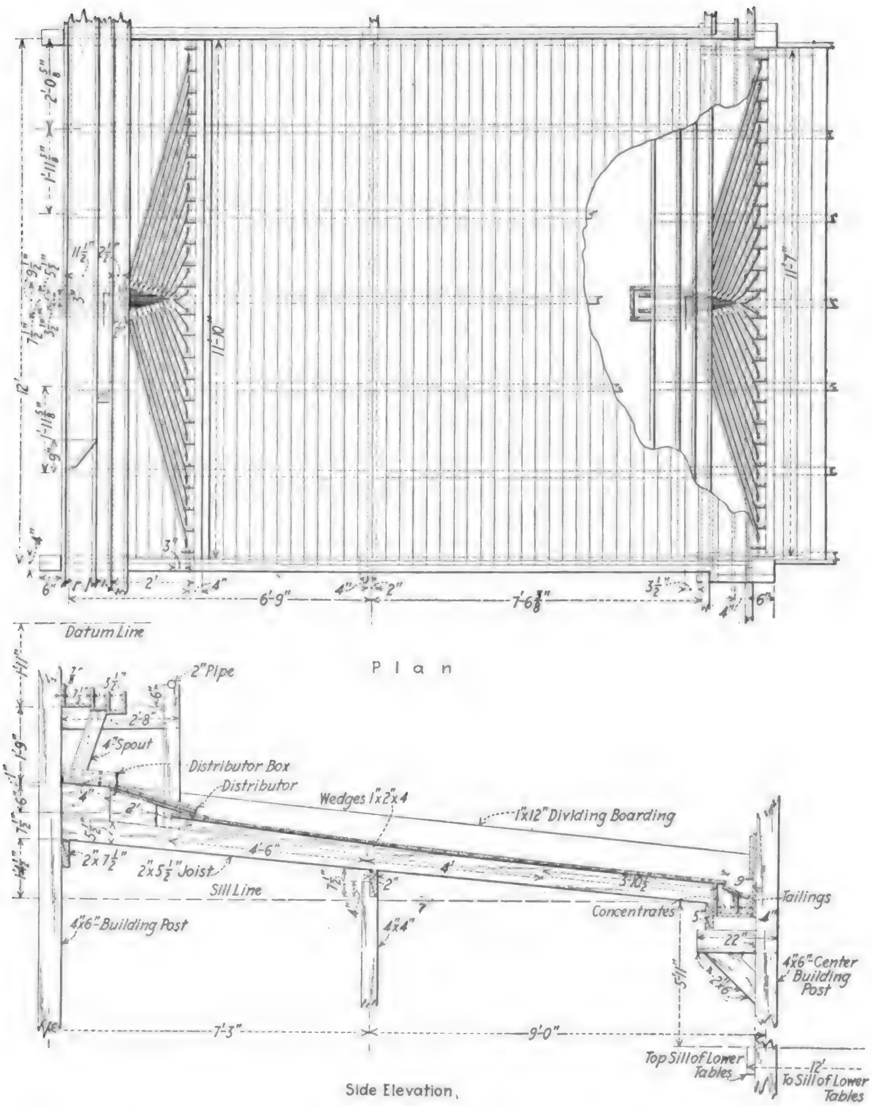


FIG. 108.—DETAILS OF CANVAS TABLES USED IN OLD COMBINATION MILL AT GOLDFIELD, NEV.

were sent to Gates vanners, and tailings from the vanners were then caused to flow over the second set of canvas tables. Still later the second set of canvas tables was abandoned.

After a canvas table has been in use for some time, especially where, as in the case of the Combination mill, lime was fed to the dry ore in the bins before crushing, the canvas covering becomes coated and the interstices of the threads filled with a deposit of lime salts. At the Combination mill it was the custom to remove this lime deposit from time to time by treating the canvas with dilute acid. In some mills where the gold is in an exceedingly fine state, this deposition of lime salts on the surface of

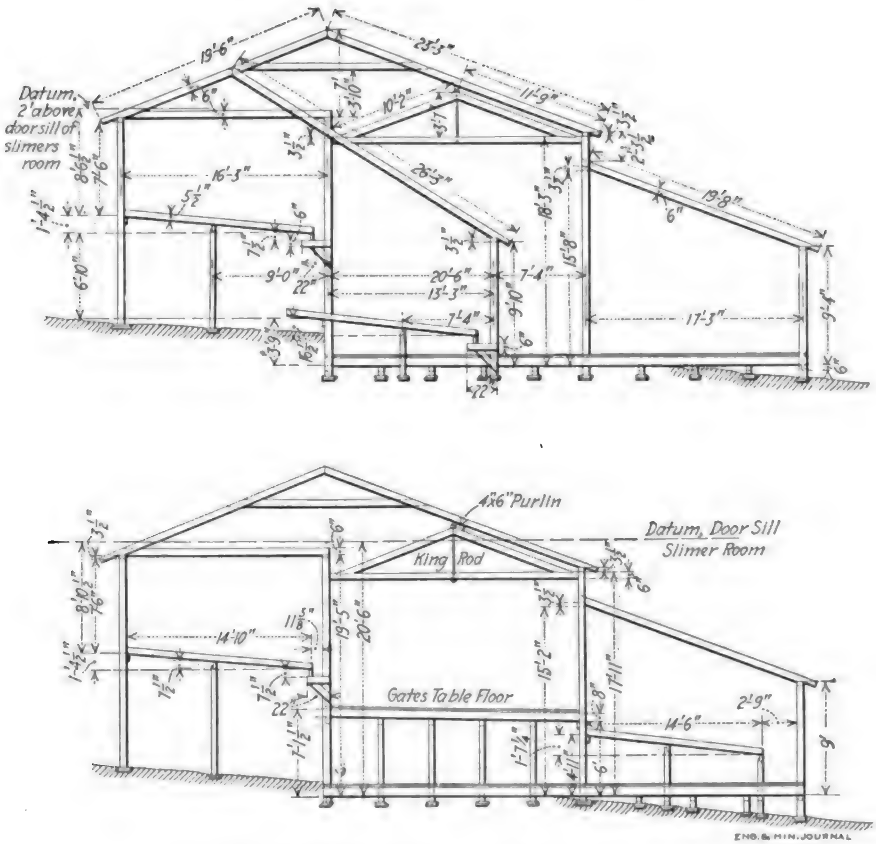


FIG. 109.—CROSS SECTION OF CANVAS-TABLE PLANT.

the canvas might not prove a disadvantage, for it often is the case that the fine flakes of gold will adhere strongly to a very smooth surface; an example being the cement tables now used in some of the Mexican gold mills. This fine gold is often efficaciously recovered by the canvas table after the pulp has passed over amalgamation tables of the usual type.

Where the amount of material caught is small, and consists largely of fine, clean, or even dirty gold, it is the practice in some mills to treat the

canvas-table concentrate in a tube mill or grinding pan, or in a Berdan pan, mercury being added and this fine or dirty gold being amalgamated. In other plants, the concentrate is cyanided, in others reconcentrated and shipped; depending on local conditions and the nature of the concentrate itself.

The amount of water that must be used to transport the finely crushed ore over the canvas surface, depends largely on the nature of the ore and grade at which the table is set. The amount of water used in cleaning is entirely dependent on the skill of the operators; some men will use three or four times the quantity of water that will be amply sufficient for another. At the Combination mill, 54% of the total ore milled, or an average of 1400 tons of slime per month, was treated in the canvas plant. The percentage of the total recovery made by concentration on vanners and canvas tables ranged from 17 to 28%; by the canvas tables alone, 7 to 13%. The average weight of concentrate recovered per month after passing over Gates vanners was 21.4 tons. The sizing test of the feed is shown in Table XXVII. The costs of operation were as follows: Labor, 18.5c.; repairs, 3.0c.; power for vanners, pumps and lights, 3.0c.; acid, 0.5c.; sundry supplies, 5.0c.; total, 30.0c., per ton of pulp treated, not including superintendence and general expense.

Table XXVII.—Sizing Analysis of Canvas Table Feed

Mesh	Heads percentage held	Tails percentage held	Extraction
+100	1.5	1.3	0.300
+150	12.2	9.8	0.600
+200	0.07	0.7	0.792
−200	13.3	9.6	
−200 (slimes)	72.5	78.4	11.310
	100.2	99.8	13.002

Saving Lead Slimes.—In the milling of lead ores a certain amount is always lost as slime. The fine particles of galena will remain suspended in the moving current of water and finally reach the tailings pile. The following scheme for collecting the lead from the slimes has been adopted by the Federal Lead Co. at Flat River, Mo. The slimes from the concentrating plant are conveyed to 18 wooden settling and dewatering tanks, 20 ft. in diameter and 20 ft. deep, with a conical bottom. In the bottom of each tank is a 3-in. pipe, shown in Fig. 110, and which passes out beneath the tank and then up about two-thirds the height of the tank. The pipe terminates in a goose neck and in the end of the pipe is a wooden plug, in which is a hole 1 in. in diameter to reduce the flow of

water. The total flow from all the spigots is 720 gal. per min. Through this pipe the heavy particles pass under a 5- to 10-ft. head and are carried by a launder to the canvas plant. The overflow of 250 gal. per min. from each of these tanks is taken to a concrete tank from which it is pumped back to the mill for re-use.

The canvas plant has eight sets of canvas-covered tables, four on each floor of the building. The tables are 12 × 14 ft. and there are six in each group. Two rows of tables are placed back to back and slope about $\frac{9}{16}$ in. per foot, as shown in Fig. 111. The tables are covered with 18-oz. canvas and it is upon this that the fine lead is collected. The slimes are run to the tables from a launder and distributed by means of a distributing trough *b*, in the bottom of which are ½-in. holes, 3 in. apart. The pulp is further distributed by passing over notched boards. The notches in each board are about 1 in. deep, 1½ in. apart, and the boards are so

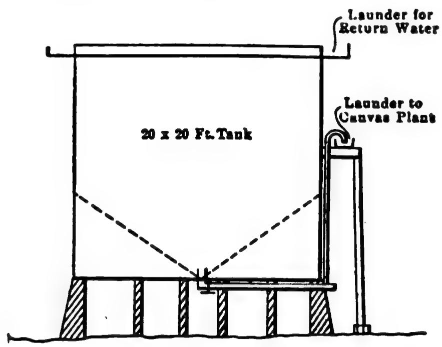


FIG. 110.—SLIME-DEWATERING TANK.

placed that the notches alternate. The slimes are allowed to flow over the tables for 30 or 40 min. Then clear water is turned on through the distributing trough for a few minutes until the barren sand is washed off, leaving the canvas covered with lead concentrate. While the clear water is flowing over the tables it has a tendency to cut channels in fine sand on the canvas. This is prevented by a boy who works barefooted, and with his foot turns the course of the water, thus preventing the lead concentrates from being washed off. The galena is then washed off by water from a nozzle and carried into a collecting tank, by the launder *d*, where it is pumped to a settling tank by a sand pump. All the work is done by boys, one boy caring for 6 to 10 tables. A screen test of the feed to the canvas plant shows: on 80 mesh, 1%; on 150 mesh, 25%; on 200 mesh, 8%; through 200 mesh, 66 per cent.

The receiving tanks for the concentrates are four in number, 10 × 16 ft., and 3 ft. deep. The lead slimes are pumped into one of these tanks and allowed to settle until the tank is full of concentrates, when the flow

is diverted to a second tank. The water from each of the four tanks flows over the edge and passes by a launder to a second settling tank, 16 ft. square, where the last of the lead is collected, and the water allowed to run off. The plant has been in operation a number of months and the management is well pleased with the results obtained. There is no machinery to get out of repair and the operating cost is low as it requires only one man and four to six boys. This is the only plant of its kind operated in southeast Missouri.

One special feature of this plant is a double launder, in one side of which,

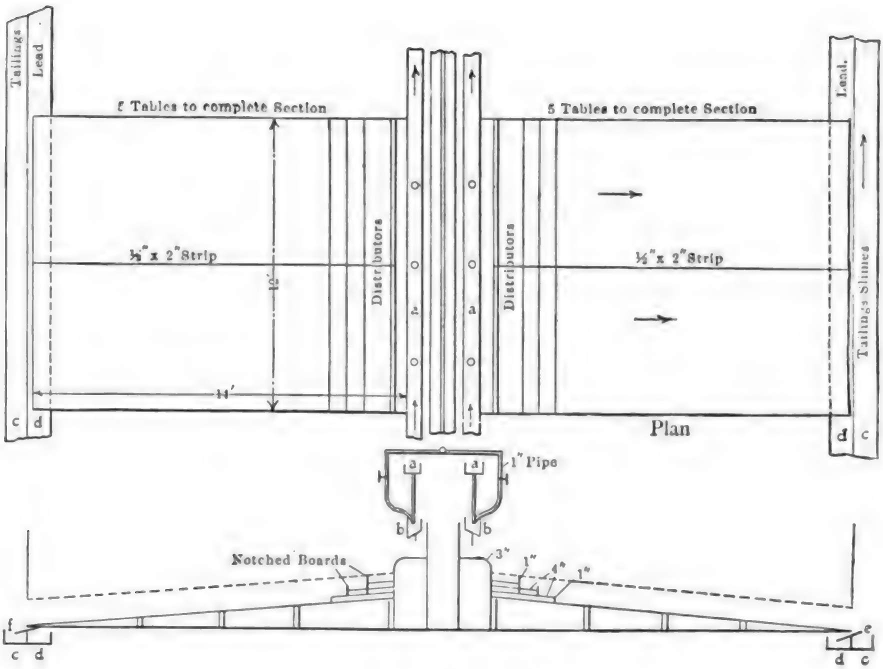


FIG. 111.—CANVAS TABLE FOR LEAD SLIMES.

c, the waste slimes are carried off while *d* carries the lead concentrates. This is accomplished by means of a board *f* about 8 in. wide, the length of which is equal to the width of the table. This board forms an apron which, while in one position, carries the slimes to the slime trough; by means of a small iron hook the board is shifted to act as an apron to conduct the lead to the other compartment of the launder, as shown at *e* in the illustration.

The Mexican Planillas.—The planilla has been used in Mexico for concentrating ores since the earliest days of mining in that country. It is particularly adapted for handling ores high in silica or lime, but heavier materials can also be treated with, of course, smaller capacity. The

details of construction are shown in Fig. 112. The slopes and ribbing are varied by the Mexicans by rules of their own, probably based on experience. In quantities of from 500 to 1000 lb., the ore after crushing and proper sizing is shoveled into the planilla, making a bed of from 3 to 6 in. in thickness. The operator stands at the lower or front end, and by means of a bowl, or horn, the water is thrown on the material in quantities to completely wet and wash it, the water both overflowing and penetrating it and passing down to the front end where it is caught, either in a pool or in a drain. If the water is caught in a pool, it is thrown over the material again. The water carries away considerable of the lighter material or waste on first washing, but it is customary to give at least three thorough washings. As the water drains through and out of

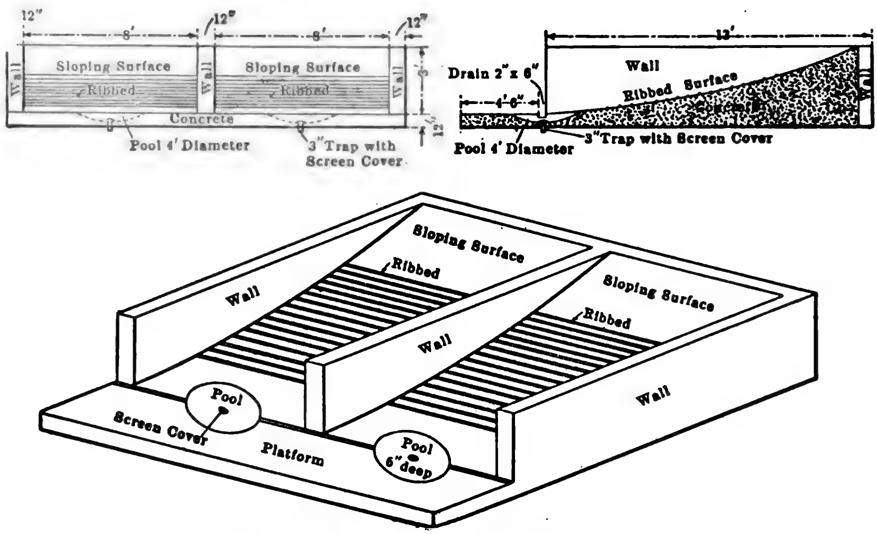


FIG. 112.—THE PLANILLA, A MEXICAN CONCENTRATOR.

the material considerable of the lighter material accumulates on top, and this is carefully removed by means of a shovel or rake. In washing, the material naturally accumulates at the lower end, but after removal of the waste it is shoveled back before re-washing. The San Robert mine treats 2 tons of headings per day, concentrating 5 to 1 from a sized material. A good extraction is reported. H. J. Baron in describing the operation states the average capacity to be about 1½ tons per day, one peon and one or two muchachos being employed. While planillas cannot be said to compare in efficiency with any of the standard tables, for a temporary device to ascertain what can be done with an ore they can be and are used to advantage by operators other than Mexicans, particularly where labor is cheap.

AMALGAMATION

Design of Plates and Tables

Area of Amalgamation Plates (By Clarence C. Semple).—Since the cyanide process for extracting gold from its ores has become so widely applicable, there has arisen a faction that advocates the abandonment of amalgamation and the adoption of the all-cyanide treatment. Amalgamation plays a minor part in the recovery of gold in the ore treated at the Goldfield Consolidated and certain other mills, but the Goldfield company did not abandon the use of the amalgamation plates in front of the batteries because of the inefficiency of amalgamation, as the all-cyanide advocates might suppose, but to prevent loss by theft of amalgam from the plates.

In the treatment of ore where an all-sliming process is used and where all the gold is in a fine state of division there would not seem to be any advantage in amalgamating before cyaniding, but where the gold is coarse the treatment of the ore by cyanide without previous amalgamation might necessitate such a long time to allow for the dissolution of the coarse gold that an inordinately large cyanide plant would be required. In such a case the removal of coarse gold by amalgamation would be an essential feature in the treatment of the ore.

However, aside from the consideration of the advantages offered in the recovery of coarse gold, amalgamation will for a long time to come continue to be an important process for recovering gold. Especially at the small mines, where the owners may not be able to afford a cyanide plant, will amalgamation plates be found whether the gold in the ore be fine or coarse. Many small cyanide plants will probably be paid for from the gold recovered by amalgamation.

An examination of the pulp that has passed over an amalgamation table will generally reveal the presence of a certain amount of amalgamable gold which has escaped amalgamation either because it has not come in contact with the mercury-coated plates or because it takes an appreciable space of time for such gold to amalgamate and in its passage over the table it has not been in contact with the mercury for a sufficient period of time. The time element also enters inasmuch as it takes an appreciable space of time for the gold particles to settle through the stream of pulp.

Given clean mercury and well kept plates, the improvement in the efficiency of amalgamation would seem to consist in giving the gold particles more opportunities to come in contact with the plates and a greater interval of time in passing over them. It is probably true that the greater the time taken in flowing over the plates the greater the number

of chances for a gold particle to come in contact with them. To do this, wide plates and a thin layer of pulp are necessary.

The Homestake mills are noted for the great area of the copper plates. The unusual length increases the time that the pulp is flowing over an amalgamated surface, but perhaps, as one engineer has pointed out, the long plates have a pronounced action in retarding or breaking the passage of the pulp particles over them, somewhat after the manner of a canvas table, except that on the canvas table the retarding action of the cloth gives the heavy particles a chance to settle between the threads. The effect of extremely long plates may be similar in kind but different in degree, and the gold particles become attached to the amalgam on the plate instead of sinking between the threads. The long tables undoubtedly do have considerable effect in overcoming the inertia of the particles hurled rather violently through the screen by the dropping stamps.

Another consideration in amalgamation is the quantity of water required and slope of the tables. The finer the ore particles the greater the quantity of water that will be required to make a thin pulp, but the pulp composed of coarse particles requires a much steeper grade to the plates than pulp composed of fine particles, or else a large additional quantity of water must be used to sluice the coarse particles over the plates. In usual mill practice the grade of the plates is greater than required by the fine pulp but less than required for the coarse. To prevent the coarse from banking, the quantity of water is increased, which greatly dilutes the fine pulp and increases the velocity of those particles in passing over the table. In other words the excess water decreases the time interval for the passage of fine particles and if additional plates are added, the coarse particles are in contact with the amalgam for an unnecessarily long interval and may even cause loss of gold by sorting.

The rational balancing of the plate area would seem to lie in using two or more sets of tables and classifying the battery discharge so as to make two or more sizes of pulp, then causing each grade of pulp to pass over the tables inclined at the most desirable slope for that size, and with the desired amount of water for each classifier product. The finest pulp would then pass over the table with least inclination, but the pulp would contain the most water. To make the interval of contact or time of flow equal for the various sizes, the finest pulp would require the shortest and the coarsest pulp the longest tables but there is a consideration other than time to be regarded, that is the opportunity or chance of a particle of gold coming in contact with the amalgam. Such chances may within reason, be assumed as approximately in inverse proportion to the number of particles in a given volume of the pulp, in other words, the finer the particles, the greater number there will be in a unit volume and the smaller the chances of any one of them reaching the amalgam, because the

finer particles will have less power to sink through the layer of pulp and because the force of the stream, which will depend upon the grade of the table, cannot be reduced beyond a certain limit without causing banking. To increase the opportunities for the fine particles coming in contact with the amalgam the time of flowing over the plates must be increased, so regarding all considerations the fine pulp should flow over the longest plate; the coarse particles over the shortest. Of course, these statements will not be true if the plates are too crowded.

It would probably be found that by using such a classification before amalgamation that the total quantity of water could be materially reduced, and with two or three sets of plates set at the least grade necessary to keep the pulp flowing over them either more gold would be recovered or a less area of plates would be required to recover the same quantity of gold as compared with amalgamation without classification.

(By W. R. Dowling).—Although there is no rule for the area of amalgamation plates there is no doubt the number and area of plates in common use on the Rand are unnecessarily large and when these are placed in a separate plate-house the total area may be materially reduced. My opinion is that much of the amalgam found at the lower end of the long battery plate is worked down by the amalgamators, who wish to present a uniformly bright surface all over the plate.

The old practice was to install plates about 15 ft. long by 5 ft. wide for each battery of five stamps, which is equal to an area of 15 sq. ft. per stamp. Where, say, one standard large tube mill for 30 stamps is erected, and assuming that the tube mill is equivalent to 30 stamps, five plates 11 × 5 ft. would probably be used. The combined plate area is then: for 30 stamps, six plates, each 15×5 ft. equal to 450 sq. ft., or 15 sq. ft. per stamp; one tube mill, equivalent to 30 stamps, five plates each 11 × 5 ft., equal to 275 sq. ft., or 9 sq. ft. per stamp; total for 60-stamp units equals 725 sq. ft. or 12 sq. ft. per stamp unit.

At the Randfontein Central the total plate area amounts to 9.6 sq. ft. per stamp unit, but it is now proposed to use only 4.8 sq. ft. per stamp, and there is little doubt that the smaller area will be quite sufficient. In the Simmer & Jack rearrangement the plate area is about 2 sq. ft. per stamp.

With the high stamp duties of recent years the same plate area is doing nearly double the work it did in the past, and is capable of doing a great deal more. Conversely, in regard to the usual 10% fall, where this is increased, more crushed material with smaller water ratio may be amalgamated with equally good results. H. W. McFarren, in his recent interesting book, says the fall of plates varies from 1½ in. to 3 in. per ft. and should not be less than 2 in. or 2½ in. A fall of 2 in. per ft. is equivalent to 16.7% and 2½ in. to 28.8% while the average is 18.75 per cent.

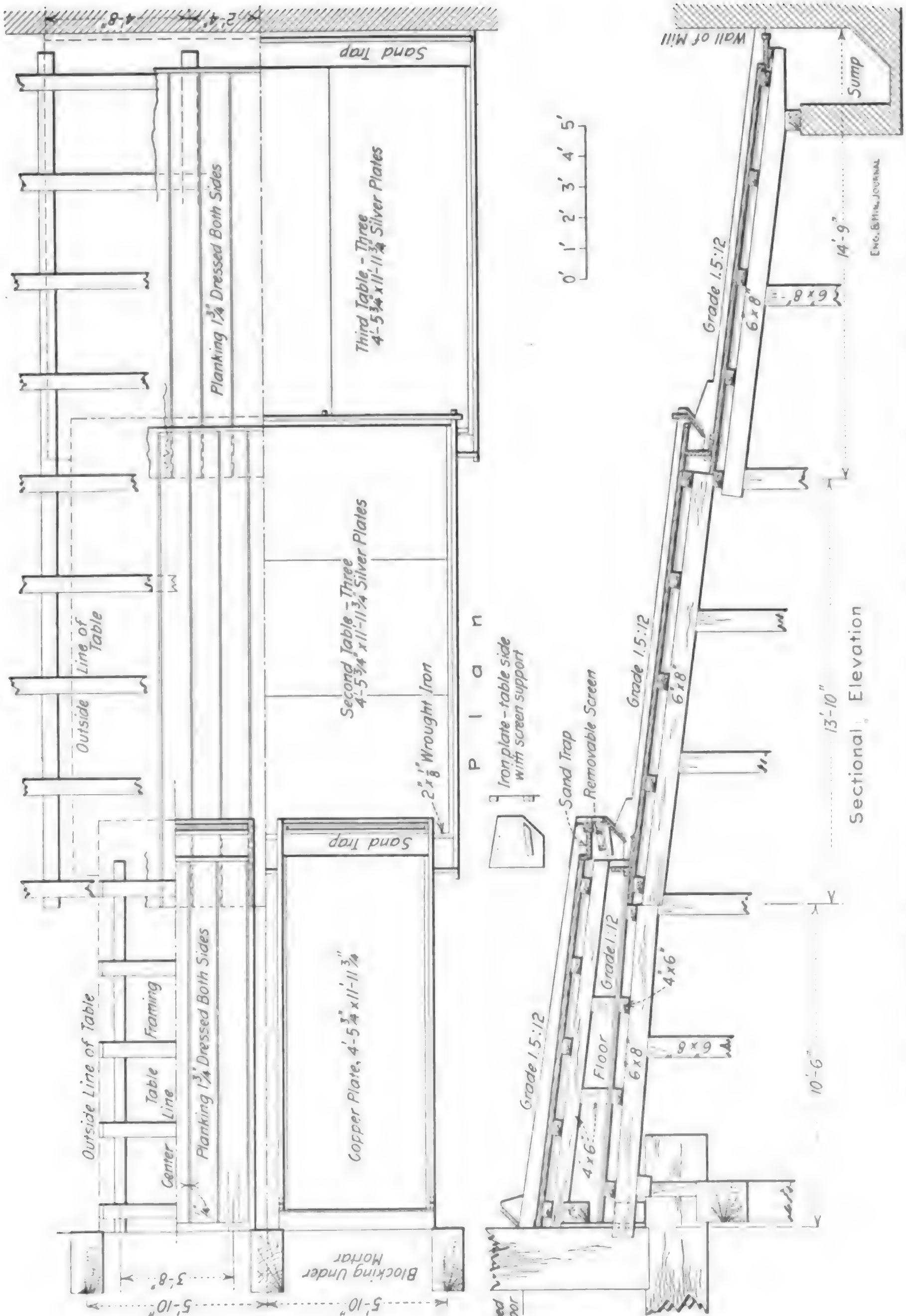


FIG. 113.—AMALGAMATION TABLES IN THE AMICUS MILL OF THE HOMESTAKE MINING CO.

Among the advantages due to reduced area of plates are lower costs of installation and operation and even more especially of mercury, the consumption being in proportion to the area of the plate. In the Simmer & Jack plants the consumption of mercury has been reduced to one-sixth of the former figure. Where a large area is exposed in amalgamation, a proportionate amount of gold is taken to set the plates, and is not available for realization until the end of the life of the plate. The gold held by the well set plates may be taken at one ounce per square foot of plate. A 200-stamp mill having 40 plates each 15 × 5 ft. will thus absorb 3000 oz. of gold.

The future points to the reduction of the use of plates to the point where only the gold too coarse for cyaniding will be caught by amalgamation. This point has probably been reached in the Simmer & Jack plant, which has now only 18 stationary plates in the tube-mill circuit in place of the former 64 battery plates and 30 shaking, tube-mill plates. It will be noticed that the Simmer East, with the plates in the stamp mill, has retained two stationary plates per tube mill, whereas the Simmer & Jack and the Simmer Deep-Jupiter joint plants have each retained three plates per tube mill. The additional plate in the last two cases is a measure for safety to have not less than two plates per tube mill in action during dressing operations.

Homestake Amalgamation Tables.—The most notable characteristic of the stamp mills of the Homestake Mining Co. in South Dakota is the enormous total area of the amalgamation plates. The company is operating six mills in which there is a total of 1000 stamps. The arrangement of the plates is not the same in all the mills, but the most approved installation is that at the Amicus mill at Lead, which is shown in Fig. 113. The three sets of tables shown in the drawing are situated in front of the batteries and are within the mill building proper. After flowing over these tables the pulp passes to the "plate house" where it flows over a fourth set of tables. It is especially to be noted that no narrow sluice- or tail-plates are used; the plates of all the lower tables are wider than the first or battery plates. The pulp from each battery of five stamps is thrown out upon a splash board whence it runs over the plain-copper plates of the first table which is 4 ft. 5¾ in. wide by 11 ft. 11¾ in. long. A sand trap and removable screen are built in the lower end of the first table. The streams from two of the first tables or from 10 stamps unite at the head of the second table. There is one of these tables for each group of 10 stamps, and are 11 ft. 11¾ in. wide by 13 ft. 5¼ in. long. From the second set the pulp passes over the third set of tables of the same dimensions as the second set thence to a sand trap and through the launder leading to the fourth set of tables in the "plate house." All plates except the first set in front of the battery are electroplated with silver, the deposit of silver

being about 2 oz. per sq. ft. The stamps crush the ore to pass a 35-mesh screen and the pulp consists of about 11 parts water to one of solids. The plates are dressed once each day. The smaller tables are brushed with an ordinary whisk broom; the larger tables with floor brooms. No scrapers are used and a thick coating of amalgam is left on the plates.

The paper on "The Metallurgy of the Homestake Ore," by Messrs. Clark and Sharwood, was discussed at the meeting of the Institution of Mining and Metallurgy on Nov. 21, 1912. Alfred James remarked on the rather extraordinary amount of amalgamated-plate surface used at the Homestake, stating that practice elsewhere at present was rather to limit than to increase the area of plates. In South Africa it was found that with finer grinding but one-quarter of the amalgamated surface formerly used was necessary. Hugh F. Marriott stated that on the Rand in recent years the question of plate design and area had been closely looked into, and for a time shaking tables were adopted on new installations. The results of long tests, however, showed that fixed plates, set at a somewhat steeper angle than formerly, answered the purpose as satisfactorily as any more complicated design and that nearly all the gold could be caught on the first three feet of the plate. In this connection Walter Broadbridge said that at the Sons of Gwalia many years ago the plates were enlarged from 16 ft. to 22 ft.; although 80% of the amalgam was collected in the first 2 ft., gold was always obtained on the last 24 in. H. A. White dwelt on the enormous area of copper plate in use at the Homestake plant. This amounts, roughly, to 12 sq. ft. per ton milled per day, and may be compared with the case of the Princess Estate, Witwatersrand gold fields, where the recovery by this process has been, for the months of September, October and November, 1912, 73.10, 73.54 and 74.13%, respectively, and therefore even higher than the excellent figures quoted, though allowance must be made for a somewhat higher grade of ore. At the Princess Estate the total plate area per ton of ore milled per day is 1.4 sq. ft., of which two-thirds is employed upon the tube-mill outflow, and the remainder upon feed-cone overflow. From the launder leading away to the cyanide plant only 20 oz. of gold were recovered after 12 months' run, though this is provided with baffle strips. The temperature of the pulp has risen as high as 110° F., but only slight improvement in amalgamation results from increased heat. The alkalinity of the mill water is kept as nearly as possible equivalent to 0.010% NaOH and it should be obvious, says Mr. White, that ill effects of temperature, where easily oxidizable sulphides are present, will be changed to advantages when the inhibiting effect of the presence of excess alkali is utilized.

Alaska-Treadwell Amalgamation Tables.—The amalgamation table illustrated in Fig. 114, is the standard design used in the five mills on Douglas Island controlled by the Alaska Treadwell Gold Mining Co. and affiliated interests. One large copper plate is used for each battery of five stamps. As compared with the Homestake tables, the Alaska-Treadwell table is wider, but shorter. The pulp from the tail-box at the

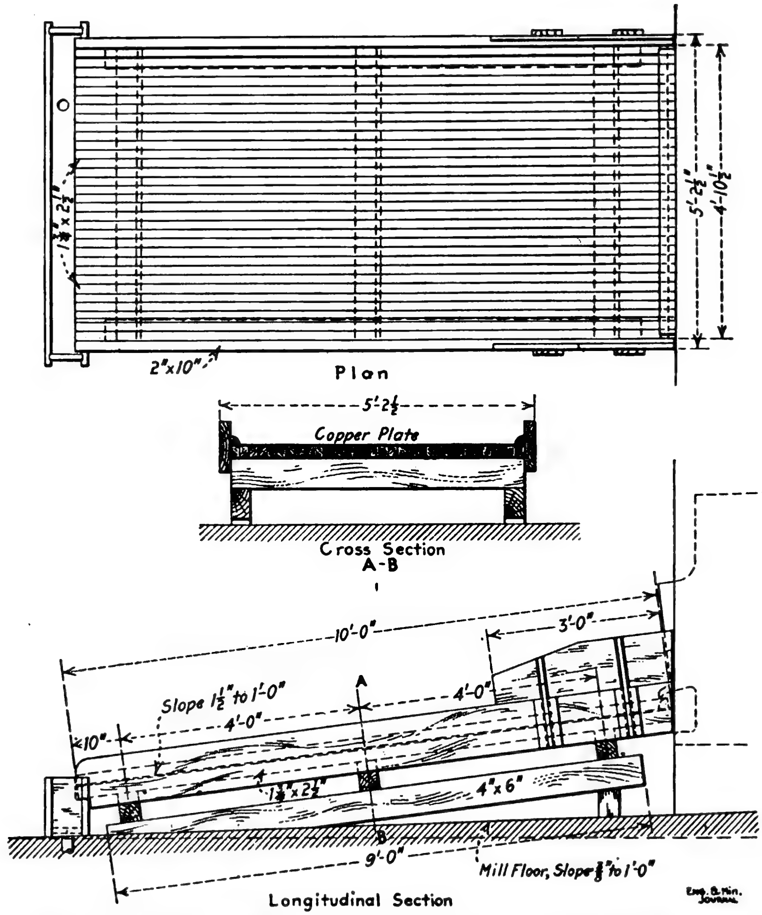


FIG. 114.—AMALGAMATION TABLES USED IN ALASKA-TREADWELL MILLS.

lower end of the table is conveyed to the concentrators. The body of the table is made up of a number of $1\frac{3}{4} \times 2\frac{1}{4}$-in. squared timbers planed smooth on the sides to make a tight joint and on the upper surface upon which the copper plate is supported. These timbers are held tightly in place by the transverse timbers on the bottom. The copper plates are held in position by the rounded cleat on each side and at the upper end of

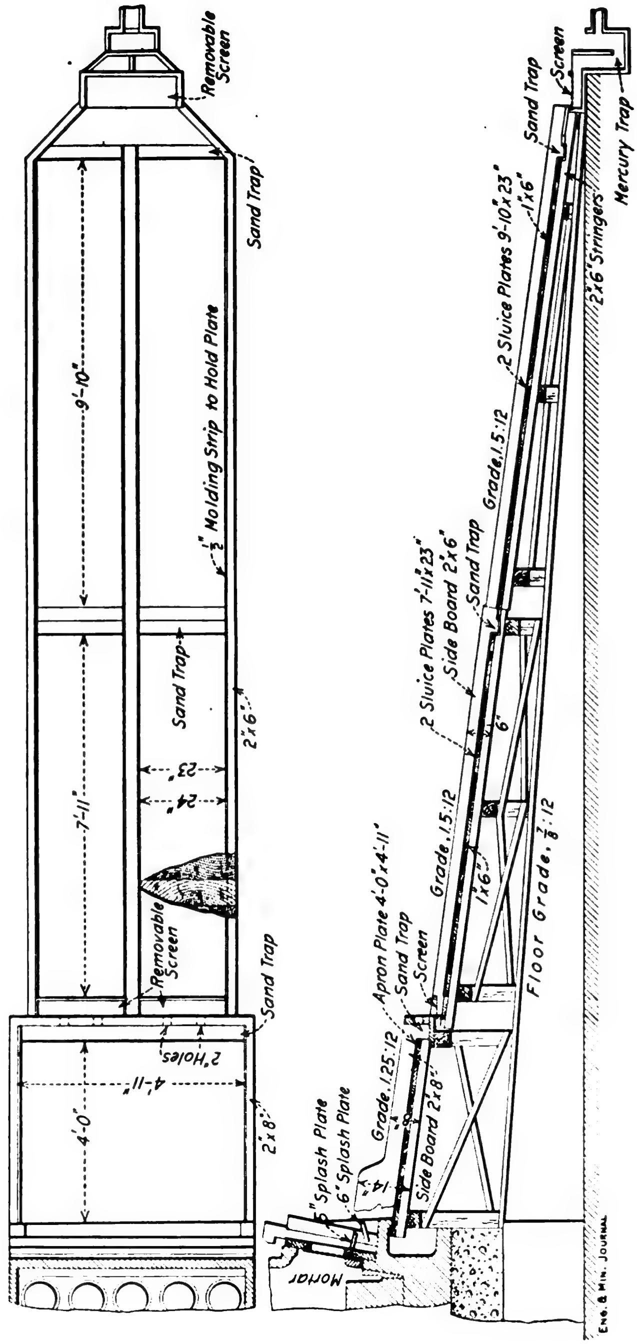


FIG. 115.—AMALGAMATION TABLES IN THE ARGONAUT MILL.

the table. The plates are dressed once a day, but if trouble from scouring, or from other causes is experienced, they are dressed as required. The operation is started by hanging up five stamps, first shutting off the feed so as to reduce the ore in the mortar to the level of the dies. The five stamps are then hung up, and a portable trough placed at the end of the plate, on the inside of the tail box. The plate is then brushed from the top toward the bottom with whisk brooms; it is then cleaned by rubbing with cloths saturated with a weak solution of cyanide. If quicksilver is required, it is now added and rubbed first with the cloth dampened with cyanide solution, and then brushed with whisk brooms at right angles to the flow of the pulp. The portable trough is then removed, some ore fed by hand into the mortar and the stamps dropped. The entire operation takes on an average 4 min. per plate. The present practice is to add very little quicksilver to the plates, except during the cleanup period which occurs once per month. During the remainder of the month the quicksilver is fed to the mortars, and the quantity so fed is sufficient to keep the plates in good condition.

Argonaut Amalgamation Tables.—The type of amalgamation table used in the mill of the Argonaut Mining Co., on the Mother Lode in California, is shown in Fig. 115. The mill was built in 1897 and the mortars were set too low. The tables are portable, are built in three sections, and can be blocked to any desired grade. It is the intention of the manager to remove the lower sluice plates, which catch only enough gold to maintain themselves in good condition without being cleaned up at all, and to use two splash plates. Experiments are now being made with these extra splash plates and the results so far obtained seem to indicate success. There are 40 stamps in the mill, each weighing 1050 lb. and crushing five tons of ore per day through 16-mesh, brass wire-cloth screens; 40 to 45% of the pulp discharged will pass a 200-mesh screen. The batteries are run with "wash" not "splash" discharge, as then the screen is not crowded, and the pulp in the mortar is thoroughly churned with the mercury that is added at the battery. This churning is conducive to good inside amalgamation, an inside chuck block being used, and while it causes a high loss of mercury better results are obtained in amalgamating inside the battery and outside. The plates are dressed once each day and only the loose amalgam is brushed off. The brushing is done with whisk brooms, no scrapers of any kind being used. A thick coat of pasty amalgam is maintained on the plates.

North Star Amalgamation Tables.—The amalgamation tables in the Central mill of the North Star Mines Co., at Grass Valley, Calif., differ from those in the North Star mill of the same company and from the tables generally used in stamp mills, in that they are built entirely of metal. The construction of the table is shown in Fig. 116. It consists

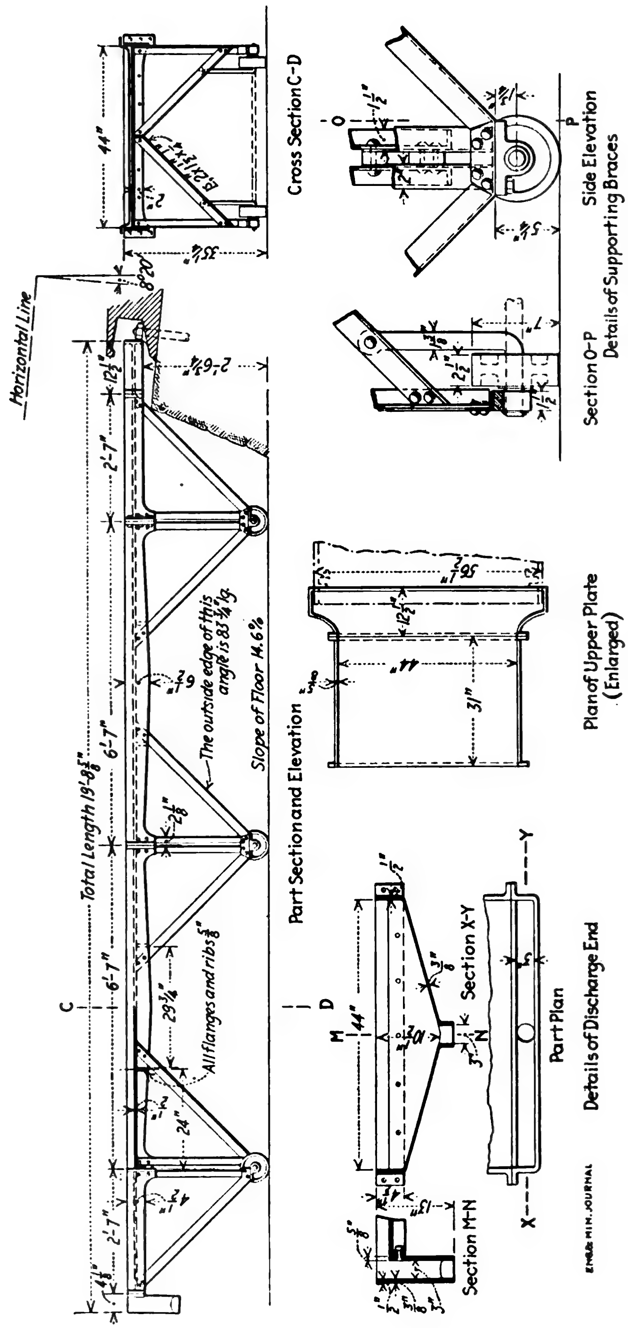


FIG. 116.—ALL-METAL AMALGAMATION TABLE USED IN CENTRAL MILL OF NORTH STAR MINES CO.

of an iron and steel support for the copper plates, equipped with legs mounted upon rollers so that the table may be moved away from in front of the battery at cleanup time or at any other time when it is desired to gain free access to the front of the batteries. The copper plates are bent upward at the sides so as to come up to the top edge of the sides of the steel support, over which the plates are again bent, in a curve of fairly large radius, so as to lie horizontally over the edge in a band about 1 or 2 in. wide. These bends along the edges are well rounded so that the amalgamator experiences no difficulty in taking off amalgam—in tables where there is a sharp corner at the sides it is almost impossible for the attendant to remove the amalgam close up against the sides of the table. At the North Star and Central mills mercury is fed to the battery at hourly intervals, the quantity being governed by the condition of a narrow strip of amalgam that is allowed to remain on the upper edge of the lip plate from one cleanup day to the next. The condition of this strip is an index to the condition of the amalgam on the chuck block. The plates are dressed once per 24 hr., a whisk broom being used for the purpose. About 12 min. is spent in dressing each table, there being one for each battery of five stamps. The work is done by the amalgamator and by the concentrator attendant, both under the direction of the mill foreman. After the daily cleanup the upper plate of each battery is treated with mercury, the quantity used depending upon the judgment of the foreman. In general 4 oz. of mercury is added at the mortar for every ounce used on the plates. About 2½ oz. of amalgam is recovered from the mortars for each ounce cleaned from the plates. While the plates in front of one battery are being cleaned the pulp is run over the plates of the adjoining stamps.

Rand Amalgamation Tables.—Since the advent of coarse crushing in practice in many of the mills on the Rand, amalgamation tables in front of the stamp batteries are not used, but shaking amalgamation tables are used in the tube-mill circuit. Before coarse crushing was adopted, however, stationary or movable amalgamation tables were used in front of the stamps. These tables were built, as shown in Fig. 117, of a series of transverse battens, held together by longitudinal tie-rods. The table thus formed was supported upon foundation posts and beams as shown in the illustration. The pulp was confined at the side, by the side-boards attached to the table bottom. Formerly it was the practice to use a number of copper plates on the table, the end of one overlapping the plate next below, but the later practice is to use but one sheet of copper on the table; this usually is about 12 ft. long and 4½ ft. wide, or slightly wider than the screen frame of the battery. In some mills the amalgamation tables are arranged so that they can be moved on rails a short distance from the mortar box, in order to gain access to the stamps, without interference

from the tables. In other mills the supports of the tables were built upon wedges so that the grade could be made greater or less as might be desired. The copper plates are turned up at the edges, in order to prevent leakage which would be liable to cause warping of the top of the table and distortion of the copper plate. A cleat or fillet of wood is nailed to the sideboard of the table to hold the copper plate in position. The plate is extended up underneath the discharge lip of the mortar. As the lip of the mortar box is equipped with a drip strip on the under edge, and the upper edge of the table is turned up, no leakage can take place. This table is somewhat similar to the amalgamation table used at the Alaska-Treadwell mills; the notable difference is that the battens

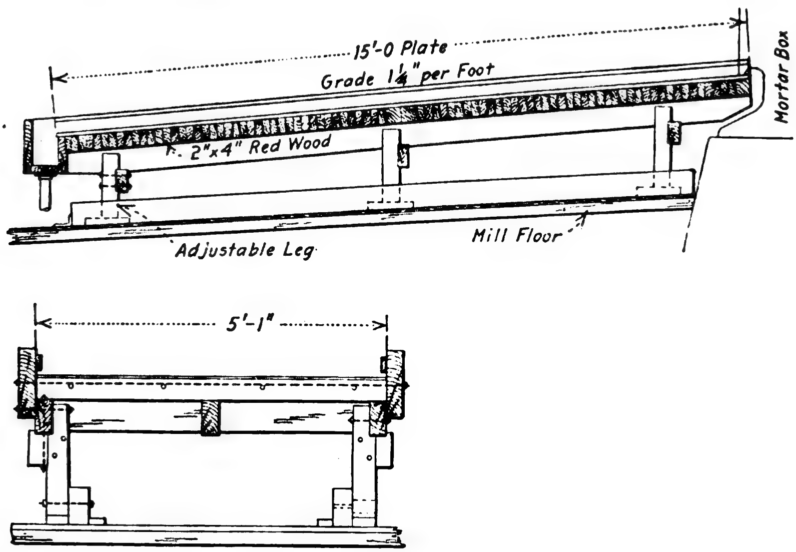


FIG. 117.—TYPE OF STATIONARY AMALGAMATION TABLE USED ON THE RAND.

instead of running longitudinally and being tied together by transverse rods, run transversely and are tied together by longitudinal rods.

Improved Rand Amalgamating Tables.—There has recently been a pronounced tendency to depart from what was long accepted as standard amalgamation practice upon the Rand. This tendency, according to C. O. Schmitt, in "A Textbook of Rand Metallurgical Practice," is greatly to reduce the length of the table, and in the newest mills on the Rand, although the weight of the stamps has been enormously increased, a much lighter table is now used than was formerly considered good practice. In Fig. 118 is shown the head end of one of these improved amalgamating tables. The copper plate is shorter and there is a steel plate at the upper end to take the wear caused by the falling pulp and to

prevent damage to the copper plate when handling stamp parts, such as heads, shoes and dies. When amalgamation was the only means of recovering gold, there was an incentive for lengthening the copper plates, and greatly extending the amalgamating area. With the gradual perfection of the cyanide process, the recovery of the more difficultly amalgamable gold has become a question of less moment. In fact, amalgamation is now essentially a process for removing the coarse gold, the solution of which, by cyanide, would greatly extend the time of treatment required.

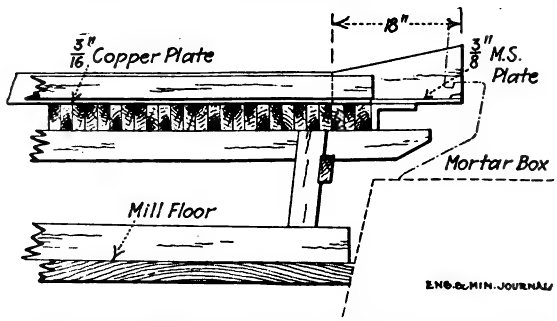


FIG. 118.—A MODIFIED RAND AMALGAMATION TABLE.

Shaking Amalgamation Table.—The amalgamation table shown in Fig. 119 consists of a comparatively small copper plate mounted upon a supporting platform in such a manner that a shaking motion at right angles to the direction of the flow of pulp can be imparted to the table by two eccentrics driven by a shaft at one side. The shaft is equipped with step pulleys, so that the speed of the shaft may be changed within fairly wide limits. The platform upon which the copper plate is mounted is made of a number of battens, smoothly planed and held tightly together by tie-rods. The platform is carried above the base by four flexible standards. The table is shown in the illustration as manufactured by the Allis-Chalmers Co., for a mill in Ontario and for several mills in Colorado, Montana and other Western mining districts. This company has also manufactured similar tables for use on the Rand, particularly to recover free gold from tube-mill pulp before it passes to the cyanide plant. The advantages of using shaking amalgamation tables has been questioned; one of the objections raised being that the mercury is jarred off the copper plate. That this objection is well founded seems to be indicated by the experience with such tables in a certain mill in the West. A narrow copper cleat was attached to the lower end of the main copper plate, and behind this cleat little balls of amalgam would roll up and find lodgment. The table is set at such a low grade that banking of the coarser particles of pulp would occur if the table were stationary. The vibration prevents actual settling and where trouble is experienced by

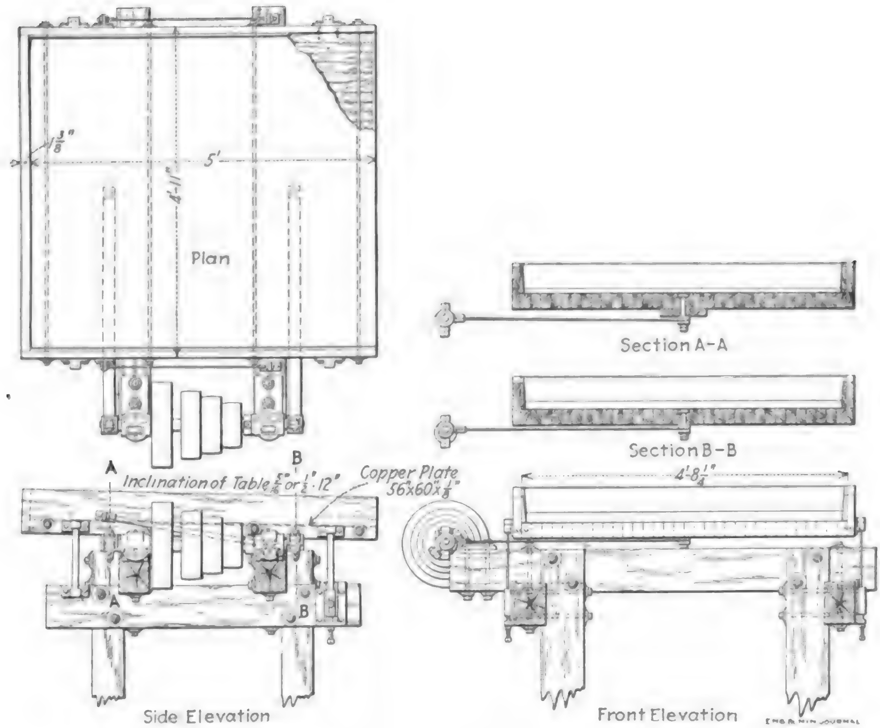


FIG. 119.—A STANDARD TYPE OF SHAKING AMALGAMATION TABLE.

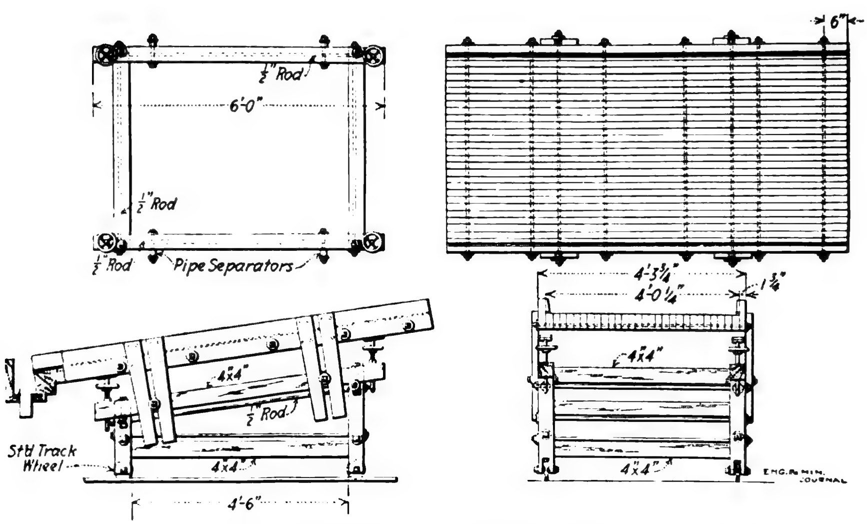


FIG. 120.—AN ADJUSTABLE AND MOVABLE AMALGAMATION TABLE.

small particles of clay settling on stationary plates in spite of heavy inclination, which settling causes the formation of a coating on the surface that interferes with amalgamation, the shaking table may be used sometimes to advantage.

Adjustable Amalgamation Table.—The amalgamation table shown in Fig. 120 differs from those described heretofore in that it is made in two parts, a table or platform for the copper plate, and a truck with wheels upon which the table proper is adjustably mounted. This construction is particularly desirable when the table is to be used directly in front of tube, chilean, Huntington mills, or grinding pans, as the plate table may be lowered to clear the discharge lip of these machines and the entire structure be moved forward on the tracks so as to make the front of the mill accessible while repairs are being made; the grade adjustment permits drawing the head end of the table tightly up under the discharge lip of the crushing machine and setting the table at any desired slope. Not the least important advantage of a movable table is that it may be so placed that no grease or oil will fall upon it while overhauling of the mill is in progress. These advantages also appertain to the use of the table in front of stamps, although perhaps not to the same degree as when it follows other crushing or grinding machines. The plate table is strongly made of a number of longitudinal $1\frac{3}{4}\times$ 4-in. battens running the entire length of the table and held tightly together by tie-rods that also hold the $1\frac{3}{4}\times$ 8-in. sides in place. The single copper plate is bound to the upper surface of the table by a cleat on each side. The plate table is supported upon the truck frame as is clearly shown in the illustration. The adjustment of the slope is made by manipulation of the four hand-wheel screws on the truck. Tables of this type made by the Allis-Chalmers Co., were installed in a mill in Eastern Nicaragua. The plate table proper may be supported upon stationary framing as in the more common practice.

Devices to Reduce Loss of Mercury

Amalgam Traps (By Percy E. Barbour).—The use of amalgam traps below the copper plates is almost universal in mills where amalgamation is practised. A great variety of traps has been designed, some of wood, others of iron, copper or pipe fittings, all of which have their merits. Referring to Fig. 121 a cast-iron trap is shown at Fig. 1 which is simple in design, effective in its operation, and which can be made in any foundry at little expense. In Fig. 2 is shown a trap made of pipe and fittings of standard sizes that is very commonly used. The only special part is the discharge lip, which can be made from light sheet iron. The trap made of pipe fittings has a decided advantage over either cast-iron or wooden traps, in that it can be made by any mechanic or other

person intelligent enough to cut and fit pipe and can be made perfectly tight against leakage.

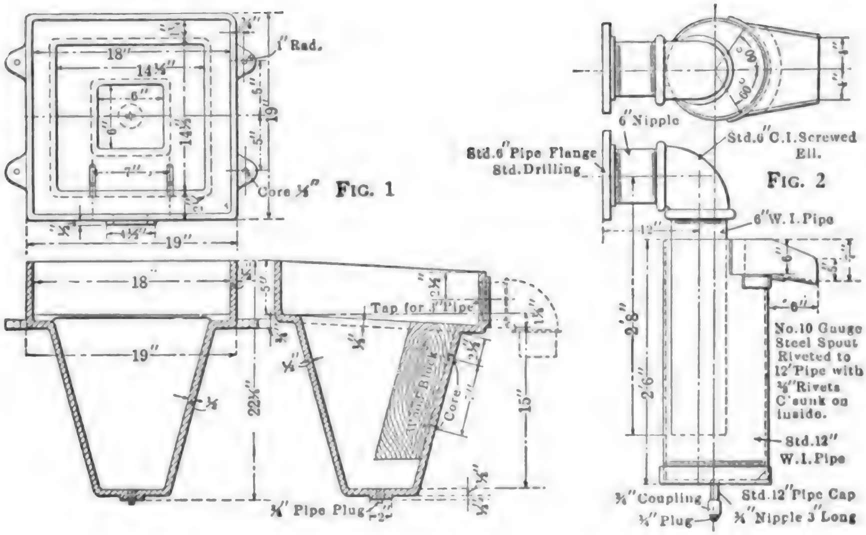


FIG. 121.—TYPES OF AMALGAM TRAPS.

Screen in Amalgam Trap.—In the Florence mill, Goldfield, Nev., amalgam traps of the type shown in Fig. 122 are used below the battery amalgamation plates. A piece of wire cloth is secured to the end of the tail plate and top of the dividing partition of the trap and serves to catch

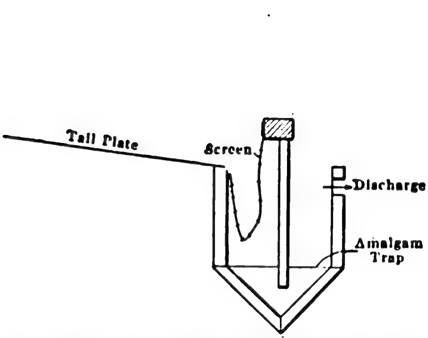


FIG. 122.—AMALGAM TRAP WITH SCREEN.

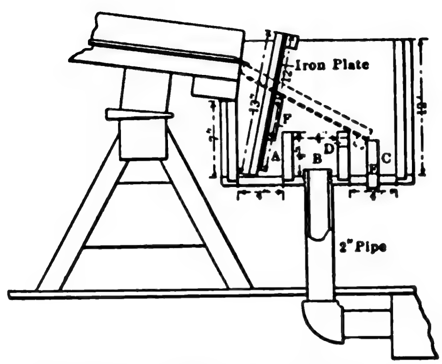


FIG. 123.—CONVENIENT AMALGAM TRAP.

lime scale, bits of wood, etc., that have passed through the battery screens. The removal of this lime scale is found to assist the later table treatment.

A Tail Box for Amalgamation Plates (By H. S. Reed, Jr.).—In order to shorten the time of dressing plates and to clean down (instead of up), as the most satisfactory method of combating the green spots that appear on the lower ends of the plates, I designed the tail box shown in Fig. 123. It consists of a box made entirely of 1-in. lumber, dressed and well jointed, a trifle longer inside than the outside width of the plate table, and 12 in. deep. It is divided into three compartments, each 4 in. deep by 4 in. wide, and contains a bafflé of No. 20 black iron. The bafflle is made by nailing a strip of iron, 12 in. wide, and of such length that the baffle will slip freely into the box without being too loose, to two end strips and one cross piece. These strips are of 1 × 1-in. wood, the end strips being 13 in. long and the cross piece having the same length as the iron plate. The end strips are fastened to one side of the iron and the cross strip to the other, the iron thus being between the strips where they join at the corners. While the stamps are dropping, the baffle is in the position shown by the heavy lines, when the compartments *A* and *B*, together with the discharge pipe screwed into and above the bottom act as a trap. The baffle rests upon the extended ends of the end strips, thus leaving a space at the bottom of the box through which the tailings may flow, and is supported by the cleats *F*, one at each end.

When ready to dress or clean the plates, the baffle is placed as shown by the dotted lines; thus, when the plates are flushed off, the loose particles of amalgam or rich concentrates are caught in compartment *C*, while the water escapes through the hole *D*. The end strips on the baffle coming outside the edges of the plate prevent amalgam or mercury from falling into *A* or *B*. When the plates are dressed an amalgam pail is placed under the box and the plug *E*, which should be long enough to support the baffle, is withdrawn, and the sludge swept from *C* into the pail. This box is not fastened to the plate table but is supported by blocks, not shown in the illustration, at each end, and is easily removed by having a union in the discharge pipe or a loose board in the floor over the tail-sluice. The elbow should be used, as otherwise there will be constant trouble from splashing and renewing the bottom of the tail sluice.

An Amalgam Trap.—A form of amalgam trap that is giving good satisfaction on sands heavy in sulphides at the Laurentian mill in Gold Rock, Ont., is illustrated in Fig. 124. The baffles are iron plates, adjustable in height above the bottom of box and sliding in grooves cut in the wood sides of the box.

Arrangement of Traps and Plates for Stamp Mill.—Though all the traps and plates shown in Fig. 125 are in use in the stamp mills of this country, yet the combination is of interest as one which gives excellent results. The series of plates, back, chuck block, splash, lip and aprons, together with the drops and the trap above the aprons, almost precludes

the possibility of escape of any particles of amalgam or amalgamable gold. Another trap is, of course, placed at the bottom of the aprons in the usual manner.

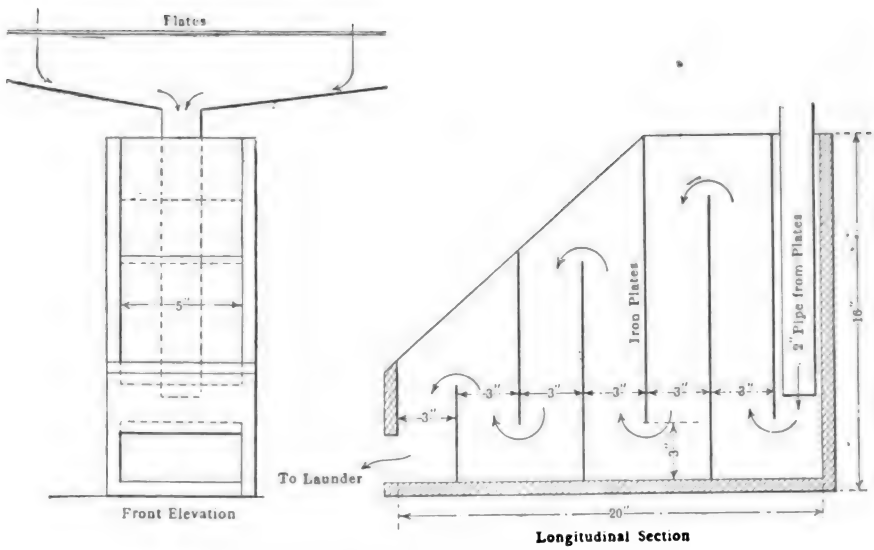


FIG. 124.—TRAP FOR COLLECTING AMALGAM.

Referring to the drawings, the pulp passes through the screen into the splash box which is suspended in front of the screen by a pair of strap-iron hangers. The pulp flows off the splash plate over the edge nearest

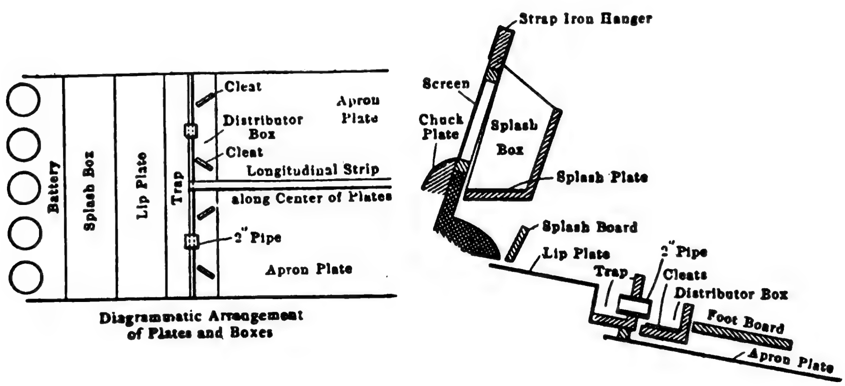


FIG. 125.—SECTION OF LAURENTIAN MILL.

the mortar, falls upon the iron of the mortar box and from there upon the lip plate. The splash board shown is to catch any flying particles. From the lip plate the pulp drops into a box or trap extending the full width of

the aprons, and from this trap it is delivered through two 2-in. nipples to the centers of two distributor boxes. These boxes are supported clear of the aprons by strips resting on the edges of the table and by the divider strip at the center of the apron plate. The cleats are tacked in as necessary to secure an even distribution of the flow. The pulp leaves the distributor boxes over the edge nearest the battery, falling upon the apron. The pulp thus impinges upon a plate three times after passing the screen, and, in addition, most of the coarser pieces of loose amalgam are held at the upper trap. On an ordinary grade of ore with this arrangement, there will be but little amalgam on the lower apron plates.

The double arrangement of apron plates is indicated in the upper drawing. With this cleat or strip extending longitudinally down the aprons, together with the trap and double distributor boxes, the whole flow of pulp may at will be deflected to one or the other side of the plates by merely putting a wooden plug into one of the 2-in. pipes leading from the trap. In this manner the necessity of hanging up the stamps while dressing the plates is avoided. The arrangement is particularly applicable to the running of low-grade ores, when the inside plates should be entirely dispensed with, and the height of discharge regulated to give the maximum capacity without regard to any scouring action that might be set up in the battery.

Notes on Operation

Methods of Fastening Amalgamating Plates at the Homestake.—It is well known that when copper amalgamation plates are fastened to the wood tables or sluices by steel nails or screws, the heads of the latter are rapidly attacked by electrolysis and rust off. In some mills, using relatively narrow tables, this difficulty is overcome by using wood cleats at the sides to retain the plates. With wide tables this is impracticable, and in the Homestake mills, at Lead, S. D., solid copper nails or spikes have been long in use, the heads becoming amalgamated at once, like the plate itself (A. J. Clark and W. J. Sharwood, *Bull.* 98, I. M. M.). These, however, are difficult to draw when replating becomes necessary, so that plates are often bent in the process of removal. Steel screws were therefore electroplated with copper, starting with a cyanide solution and then using the sulphate as electrolyte. The heads of these become amalgamated in the mill, but the metal is protected, so that they can be unscrewed after over two years' use. They promise to be a cheap and effective substitute for the copper nails.

Device to Prevent Scouring of Amalgamation Plates.—Especially where coarse crushing is practised there is usually an excessive scouring of the amalgamation plates where the pulp flows from the distributor to the plates. In the Florence Goldfield mill this scouring action is prevented

by placing iron plates, ¼ in. thick and 1½ in. wide, across the top of the plates. The pulp falls or flows on these iron strips from the distributor and then to the amalgamation plates. By this simple means a great saving in plates has been effected at practically no extra cost. In cleaning the plates the strips are simply lifted out and the plates dressed as usual.

Dressing Plates without Hanging Up.—A system which permits the dressing of plates without hanging up the stamps is in use on the Langlaagte Estate, South Africa. It consists of placing a board across the top of the table which diverts the pulp over the back of the table to a launder which runs beneath the plate. This launder carries the pulp to a separate plate table kept for the purpose. Ordinarily the pulp is prevented from flowing over the back of the table by a strip of wood, the pulp overflowing this strip when the "dam" is put in place.

Verdigris on Amalgamation Plates.—The stains of verdigris that frequently appear on amalgamation plates consist of oxide and carbonate of copper. The copper in the amalgam is oxidized by air and water. The copper may have been amalgamated with the gold, its salts may have contaminated the mercury, the pulp may contain compounds of copper that oxidize, chemicals may have been used in dressing the plates, but ordinarily verdigris stains result from there being too thin a coating of amalgam on the plates.

The stains are readily soluble in potassium cyanide and acid solutions, but as these chemicals also attack the metal of the plate, their use is always attended by reappearance of the stains. The best remedy is to leave more amalgam on the plates at cleanup time until a thicker coat has accumulated. To build the coat of amalgam on the spotted areas, brush them lightly with a 2% solution of potassium cyanide; then when the stains disappear, wash with water until all the cyanide solution has been removed. By persistent rubbing, at cleanup times, from the edges of the stained area inward, the amalgam can in time be brought to thickly cover those areas. Silver-plated plates are less apt to become stained than plates of plain copper. A little sodium amalgam often helps matters, but its use is not attended by beneficial results if used too liberally.

The mercury should be cleaned by retorting, then washing in 10% nitric acid solution, stirring frequently. The acid which dissolves the base metals and some of the mercury should be siphoned off for use again. The mercury must be washed free from the acid before being used.

(By Algernon Del Mar).—The preventive in this case is not to use chemicals of any sort on the plates but to use a wash of borax soap. I learned this from Courtney De Kalb at the Exposed Treasure mill and since I have made use of borax soap I have had no difficulty with verdigris. Even with plates that have been bared of the silver plating and where

verdigris ordinarily will appear I find by washing or scrubbing the plates every shift with a solution of borax soap dissolved in hot water that the plate readily amalgamates and the soap keeps the air away from the surface of the plate so that when the water is turned on the plate is bright. I used this method for two years at one mill and always had the plates bright from top to bottom. I remember some years ago I had to amalgamate in a mill where the plates were in poor condition, verdigris showing all over. The trick was to brush up the plates quickly all over at the finish and make haste to turn the water on for the verdigris came to the surface almost as soon as the brush left the plate. The management insisted on keeping the plates stripped of amalgam and depended upon the millman to find a means of keeping the plates in good condition.

Inside Amalgamation (By K. C. Parrish).—Inside amalgamation is often advisable when there is difficulty with the outside plates on which a scum sometimes forms in spite of all precautions. Amalgam is much safer and less apt to be lost in ordinary or careless work when caught on the outside but there are some cases where gold can be caught on the inside of the mortar when it cannot be amalgamated on the outside. Several years ago I had occasion to make some mill runs on several hundred tons of high-grade ore in Colombia, South America. The ore contained about 13% sulphides, half of which was pyrite and the rest zinc and lead sulphides in about equal proportions. It was found that a dark-colored scum formed on the outside plates within a few minutes after starting the mill; the addition of chemicals did not help matters. The outside plates were placed very steep so as to clear themselves of the sulphides as quickly as possible. Frequent and careful dressing of the silver apron plates had no special effect; different degrees of hardness, cleaner mercury, wells, and traps did not improve the extraction. The gold was mostly fine and part of it had to be brightened before it would amalgamate. This was done by using a higher discharge and finer screen. It was noticed that if the gold did not amalgamate on the inside but a small proportion of it could be caught on the outside plates. The outside plates could not be kept free from the scum and even the amalgam that was put on or occasionally caught, would sicken and scour off. Careful watching and feeling of the plates and the more frequent addition of quicksilver (as often as every 15 min.) raised the extraction from a total of about 50 to 70%. This extra saving was effected entirely on the inside plates. The advantage of inside-plate amalgamation is that the plates are always kept clean and bright by the splash of the pulp. Sufficient quicksilver must be added to the mortar to take up all the gold, but not so much as to soften the plates. It is best added frequently in small amounts. Some amalgam I have found to be dirty and brittle, even from the inside plates, so that with the addition of a little too much quick-

silver the points and sharp ridges break off. Only a small portion of such an amalgam can be easily caught again on the plates after it has loosened and all of it will not settle around the dies; consequently, extra care must be taken in the addition of the mercury. The objection to inside amalgamation arises from the fact that it requires considerable experience and skill to judge if the inside plates are in condition by the appearance of the head of the outside and the feel of the inside plates.

Removing Silver Coating from Copper Plates.—All copper amalgamation plates used in the mills of the Homestake Mining Co. at Lead, S. D., are coated with 2 oz. of silver per sq. ft., except the first set below the batteries, which are plain copper plates. Occasionally it is necessary to renew the silver coating, which cannot be well done without first removing the old coating and the hard amalgam adhering to the plate. The silver plating is done in the electrolytic-plating room in the assay-office building. The old coating and amalgam is removed by buffing wheels of canvas to the periphery of which steel shot is glued. The buffs are 10- or 12-in. diameter by 2- or 3-in. face and are made by stitching together the requisite number of disks cut from canvas. They are provided with a wood center pierced by a hole for the arbor of the rotating gear. The arbor is supported by a steel frame and is belt driven by a motor. The frame is so hinged as to permit an up and down motion, and so mounted that it may be swung horizontally, the driving motor being carried on the frame so that the tension on the belt is not changed by shifting the buff to any position. The copper plates are supported upon a table of planks on carpenter horses and are shifted about to bring all parts within the radius of the frame as it is swung about the vertical axis. The loosened coating and amalgam is drawn through a canvas pipe, the opening of which is immediately above the buff, by an exhaust fan, and is led through a small tank of water where it settles. The sediment is from time to time removed from the tank, dried, the shot dust removed by a magnet and the remainder retorted, the valuable metals being recovered by melting with the precipitate from the presses of the cyanide plants. The plates are not grooved or scratched by the steel shot, as is the case with emery. After being straightened they are coated on the back with varnish and placed in the electrolytic vats.

Gold Absorption by Amalgamation Plates.—The question of the absorption of gold by amalgamation plates has again been taken up by G. H. Stanley and M. P. Murray (*Journ.* Chem., Met. and Min. Soc. of So. Africa, December, 1911). The plate examined by the authors had been in use in one of the mills of the Crown mine for 12 years, and was about 0.14 in. thick. It had been sweated and scraped, but not sealed, before being discarded. The under surface was also amalgamated. A portion of the plate, on assaying, showed the presence of 1.69% gold and

1.31% mercury. Successive layers 0.02 in. thick were taken off by a planer, particular care being taken to free the edges by filing from any contamination due to the salting of the edge by amalgam working over. The seven portions ran as follows in gold: 10.48%; 0.0166; 0.00505; 0.0084; 0.0084; 0.0041; 0.0145; 0.0018%. The first layer contained 9.39 % of mercury; the last, the lower amalgamated surface, 1.89%. Mercury was also present in all other cases, but in amounts too small for determination. This shows that over 99% of the total gold was contained in the first layer of 0.02 in. From microscopic examination it appears that the gold which existed on the interior of the plate, had not truly diffused into it, but that the mercury-gold amalgam had simply entered into minute blow-holes existing in the original plate. However, the amount apparently held by the interior of the plate amounted to only about 0.25 oz., while the total gold for the plate amounted to about 46 oz. On the basis of these experiments, true absorption is negligible, and the removal of the surface to a depth of 0.02 in. would give all the gold, which, from a practicable point of view, would be worth recovering. This could be effected by the use of a large planing machine or by careful scaling or by several repetitions of sweating and scraping with hard scrapers with a fresh addition of mercury between each scraping. This latter method of cleaning recommends itself for use between short custom runs.

MISCELLANEOUS PROCESSES OF SEPARATION

Magnetic

Magnet for Removing Steel from Ore.—The details of a magnet that is used by the Federal Lead Co., Flat River, Mo., to pick pieces of steel and iron out of the ore as it passes by belt conveyor from the crusher to the rolls are shown in Fig. 126. The metal to be removed consists of pieces of drill steel, bolts, track spikes and castings from the machine drills. In fact any iron that may get in the ore on its way from the stope to the mill.

The magnet used consists of a cast-iron core, *a*, 4 in. thick, 20 in. high and 2 in. wide. It is wound with 19 layers of No. 10 double cotton-covered copper wire *c*, with 2300 turns on each pole. The current used is 5 amp. at 125 volts. The pole faces *b* are 2 × 6 × 24 in. and are spaced 6 in. apart. The magnet is suspended from a carriage *e* and supported on a track *d* at right angles to the belt. When a number of pieces of iron are collected on the magnet, the entire apparatus is moved to one side, the current cut off, and all the iron is dropped to the floor. The magnet is suspended 6 to 8 in. above the belt and is adjustable by means of turnbuckles *f*. This magnet will pick up pieces of steel weighing as much as 10 lb.; this prevents trouble and breakage at the rolls. A similar but smaller magnet is used by the Doe Run Lead Company.

The Premier diamond mine handles over 30,000 tons of material per day which contains more or less metallic iron as a waste product from the tools used. The concentration methods are so highly developed that about 1 ton of concentrates per day contains the entire diamond output. The concentrates are fed by hand into a hopper which distributes them over a slowly moving canvas belt 4 ft. wide; on this belt they first pass under a narrow magnet placed across the width of the belt, and along the face of this magnet runs a small canvas belt. All metallic iron particles

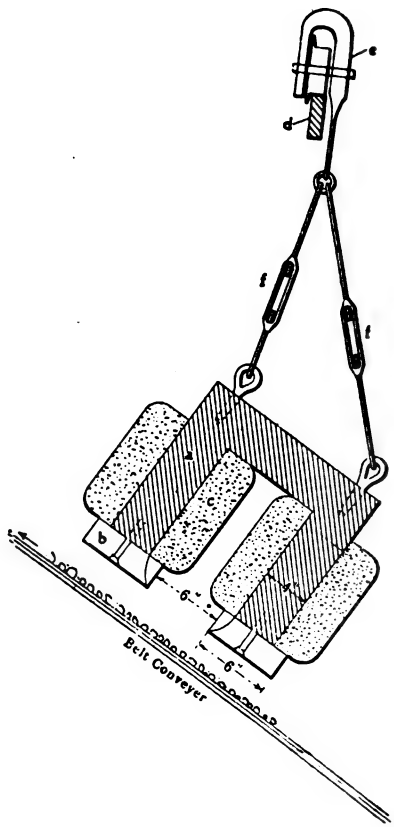


FIG. 126.—MAGNET FOR PICKING STEEL FROM ORE.

are attracted and lifted up by this magnet and carried and thrown on one side by the small belt. They amount to about 10% by weight of concentrates. The remainder is then carried on by the main belt, and allowed to fall between the poles of a pair of powerful magnets, where they are divided by a knife edge; the non-magnetic particles dropping straight down and the others being deflected. The field which produces the second sepa-

ration is an exceedingly powerful one (B 20,000 per sq. cm. estimated). The pole pieces are about 4 to 5 ft. wide, and are wedge shaped, tapering from 6 to 8 in. to a curve of about ¼-in. radius, at which part, of course, the flux is concentrated. The main coils are four in number, each resembling one of the field coils of a large two-pole dynamo. The current taken by these is about 40 amp. at 500 volts.

A closed tube-mill circuit is part of the system of cyaniding concentrate at the mill of the Alaska Treadwell Gold Mining Co., Douglas Island, Alaska. Unless some method is provided for removing iron from this circuit, it will accumulate, reaching as much as 15% of the total. To avoid this difficulty a magnetic device has been installed, which continuously removes all metallic iron from the pulp. In the launder which

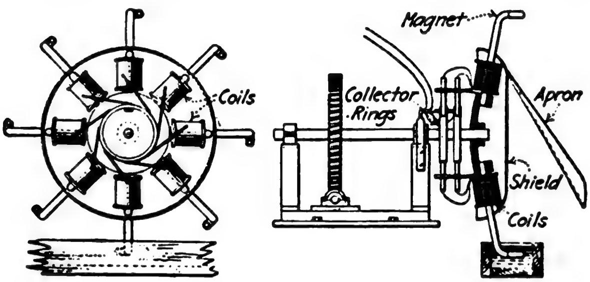


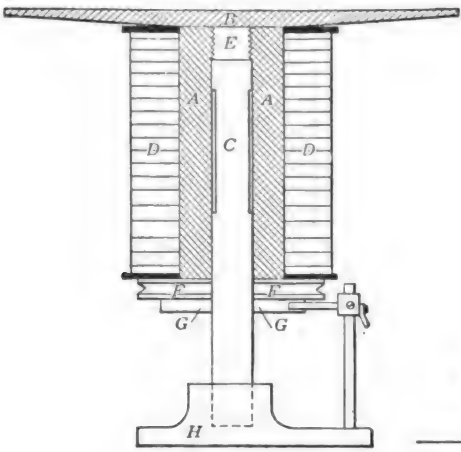
FIG. 127.—MAGNET FOR REMOVING IRON FROM PULP.

conveys the ground pulp back to the Dorr classifier for separation, a wheel, which consists of a number of magnets, is arranged; the ends of the magnets dip into the launder and remove the adhering iron, discharging it on an apron which carries it clear of the launder. Current is supplied through a simple commutator. Fig. 127 shows the details of the arrangement.

(By W. C. Brown[1]).—The following details and sketches show a machine I have designed to remove iron from stamp-mill pulp. Referring to the drawing to the left in Fig. 128, *A* is a cast-iron core, 10¼ in. long by 6 in. diameter, with hollow in center 2 in. diameter. *B* is a circular pole, face 20 in. diameter, turned perfectly flat on top, the under side being tapered gradually from the edge, which is ¼ in. thick on the top of the core, where it is ½ in. thick. It is secured to the core by four ½-in. countersunk screws. The exciting coil *D* consists of 5120 turns of No. 18 copper wire, wound on a bobbin made of sheet iron with brass flanges. The whole is made water tight with a zinc covering soldered to the iron

[1] Excerpts from an article in *Trans.* So. Afr. Inst. Engrs., May, 1910.

bobbin. One end of the coil is grounded, and the other end brought out to a slip ring *G*. Between the slip ring and core a fiber insulator *F* is placed, to which the slip ring is secured. The insulator *F* is grooved as shown, so that it can be used as a pulley to revolve the magnet. In the top of the core a brass plug *E* is screwed. This serves as a bearing, and the whole is revolved on the spindle *C*, which is secured in a cast-iron foot *H*. The exciting current of this magnet is 2½ amp. at 100 volts, direct current being used. The magnet revolves at 6 r.p.m. and is driven through a worm gear by a ½-h.p. motor. A ⅜-in. cotton rope passes over the pulley *F* and over a grooved pulley on the worm gear, a suitable tightener being fitted to take up the slack. The drawing to the right shows the position of the magnet in relation



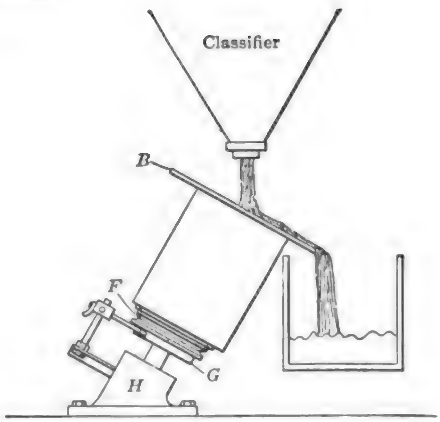


FIG. 128.—BROWN ELECTROMAGNET FOR REMOVING IRON PARTICLES FROM PULP.

to the classifier under which it is placed. The pole plate is at an angle of 35 deg. The pulp falls on the center of the plate and flows in a widening stream to the bottom edge. The iron particles are all caught on the edge of the plate and scraped off by a piece of canvas belting. No iron of any size passes the magnet, and only the very finest is lost. This is due to the fine iron not being able to come in contact with the plate and being carried away by the heavy pulp.

Magnetic Separator in Tube-mill Circuit.—The quantity of small particles of iron that may enter into a closed tube-mill circuit originating from attrition within the mill, soon amounts to a large total, unless some means of eliminating it as fast as dicharged from the tube mill is provided. The material used within the tube mill, and which becomes subject to attrition, is of the toughest of steels, and, therefore, resists grinding action within the mill to such an extent, that it is repeatedly

passed through the circuit, before being ground fine enough to be carried into the overflow product of the cones. In order to remove the accumulation of coarse metallic iron particles, a magnetic separator has been used in the Simmer & Jack mill on the Rand, and, according to C. O. Schmitt, in "A Textbook of Rand Metallurgical Practice, Vol. II," as much as 900 lb. per day of metallic iron has been removed from the tube-mill circuit. This amount is nearly equal to the whole consumption of

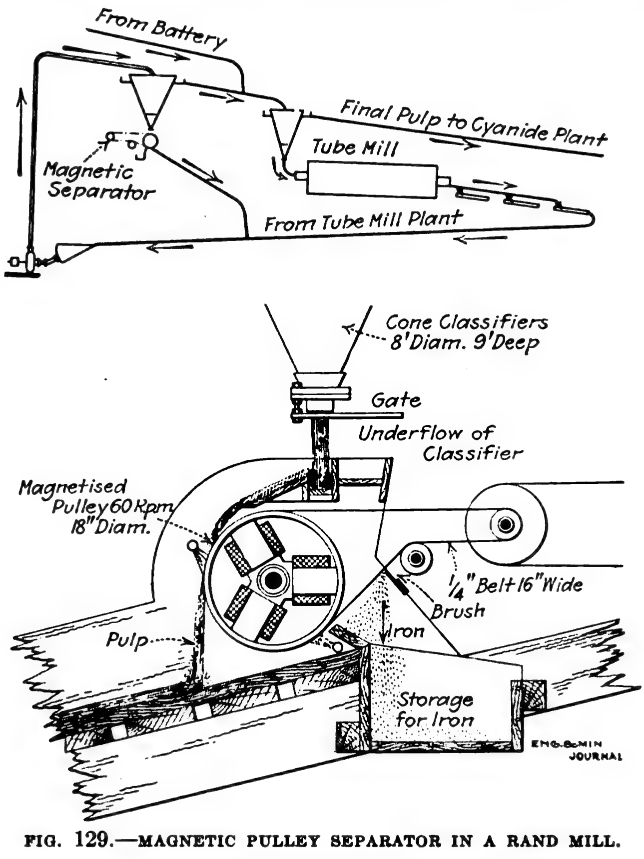


FIG. 129.—MAGNETIC PULLEY SEPARATOR IN A RAND MILL.

about 0.4 lb. of steel per ton of ore milled per day. The details of the magnetic separator and its position in the tube-mill circuit, are shown in Fig. 129. Before this magnetic separator was adopted at the Simmer & Jack mill, the accumulation of iron from the continuous wear of shoes, dies, liner plates and other parts in the crushing plant, became so great that it was found necessary to pass regularly once per day, for a period of about 15 min., the whole pulp to the cyanide plant without recrushing in the tube mills, in order to clear the tube-mill circuit of the accumulation of iron.

Pneumatic

A Dry Concentrator for Placer Gold[1] (By J. V. Richards).—At Las Palomas, Sonora, Mex. a dry concentrator or air-blast machine working by virtue of the difference in density of metal and gangue is used to recover gold from the disintegrated binding material of cemented gravel. The dry washer consists of a small table about 4 ft. long by 2 ft. wide with a cloth top and a bellows below. The gravel is fed, through a hopper, at one end. The top of the table consists of a fine wire netting like ordinary fly screen. On this is placed one and sometimes two layers of burlap, and over this a layer of thin cotton cloth; old flour-sacks are often used. Four ½-in. riffles are fastened at equal intervals across the surface. Detachable wooden sides are used to keep the gravel from spilling over. The machine is usually built to stand level, but is set on the edge of a dump at an angle of about 20° from the horizontal. It is kept carefully leveled in cross-section. A hand-crank belted to a small drive wheel operates the bellows, which has the ordinary clap valves in the bottom and canvas sides. The intermittent puffs of air from this bellows, coming through the cloth top at the rate of about 150 per min., cause the sand and gravel to jump the riffles and travel down the table and over the end. Most of the gold, together with a considerable amount of black sand, is caught behind the first two riffles. The capacity of such a machine is from 2 to 2½ cu. yd. per hour.

After several yards of gravel have been passed over the table, the machine is stopped, the side boards removed and the material behind the riffles brushed into a gold pan. It is then fed back slowly, nearly all the gold this time remaining behind the first riffle. The first and second riffles are then brushed out clean and the resulting sand, black sand and gold is "tossed" in a batea and the gold separated dry and clean. For the successful operation of the dry concentrator, it is essential that the sand be absolutely dry. Even a small amount of dampness tends to clog the cloth and the riffles. In the case of a cement gravel it is also, of course, necessary that the material fed to the machine be thoroughly pulverized. Hand pulverizing being exceedingly tedious, several disintegrating machines have been devised for this work. Most of these machines are faulty in the respect that they not only pulverize the cement but smash up the pebbles as well, which is obviously useless labor as all that is required is to clean and eliminate the gravel and pulverize the cement. The machine known as the Quinner disintegrator described elsewhere in this volume does this work more or less successfully.

Dust Collector for Dry Concentrator.—At the property of the Mount Summit Ore Corporation, near Garrison, N. Y., a suction blower has been

[1] Abstracted from an article in *Bull.*, A. I. M. E., April, 1911.

installed to take the dust out of the crushed ore at various points. The dust is discharged from the blower into the device illustrated in Fig. 130. This is a dust catcher of the vortex type, usually applied to planing mills, lumber mills, etc. This is probably the first instance of its use in a dry concentrating plant. The air and dust from the blower enter through the tangential pipe and take a circular path around a collar *A*. This

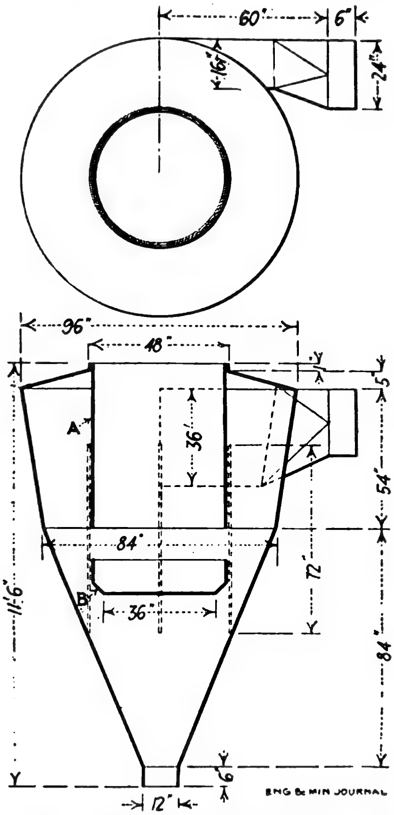


FIG. 130.—PLAN AND VERTICAL SECTION OF DUST COLLECTOR.

collar is open at the bottom and the top. The air enters it at the bottom and passes out through it; the dust settles to the bottom of the cone, whence it is removed through a bottom gate, as desired. A dished plate *B* under the collar can be adjusted for height, thus regulating the size of the opening through which the air is discharged. The device is made of galvanized iron and was assembled on the spot by the Buffalo Forge Co. It is reported to be satisfactory in service.

IV

ACCESSORY APPARATUS FOR ORE DRESSING

BINS AND DISCHARGE GATES

Storage Ore Bins

Surface Bins Excavated in Rock.—For the new Hardenberg mill in Amador County, Calif., a novel system of ore storage is employed. The collar of the shaft is 62 ft. above the top of the mill ore bin. The ore supply is trammed to the mill by hand. From the surface track leading to the mill a tunnel has been driven into the hill through the bedrock to a point about 20 ft. back of the shaft to connect with a storage excavation. The connection is made by an upraise from the bottom of the tunnel and alongside the shaft. The tunnel is 8 ft. high. The bottom of the excavation is on a plane with the top of the tunnel. This gives 54 ft. of depth available for the storage of ore between the collar of the shaft and the tunnel. This excavation is cut into the solid greenstone and has a capacity of 1250 tons of ore, which may be increased by widening the cut. A large storage compartment consists of a glory hole in the slope of the hill above the mill, which has a capacity of 3000 tons of ore. This will be made available when desired by driving another tunnel into the hill from the top of the glory hole to a point at the main or regular storage compartment, where the ore can be drawn off and trammed to the glory hole when there is an excess of ore.

Calumet Rock Chute (By Claude T. Rice).—The type of chute and gates used on the bins at the rock-houses of the Calumet & Hecla company is shown in Fig. 131. The ore that goes to these bins is crushed to 4 in. so no large pieces pass through the chute, which is used for drawing the ore off from the bottom of the bin. Liners are used to protect the bottom of the chute. These liners and the gates wear quite thin before they have to be replaced. The gates are easily changed, for by closing either one of the two the ore can be held back while a new one is substituted for the other. The bottom door or gate shuts off the ore when lifted up from below, while the other gate, which drops down, is used only to regulate the flow of ore by choking the mouth. Levers fitted to the square ends of the gate shafts engage the notched arms that are carried down from the bottom beams of the bin. In this way either gate

can be locked shut or at intermediate points of closing so as to regulate the flow of the ore. The chutes are usually built to discharge at right angles with the railroad tracks. Where the chutes discharge parallel to the

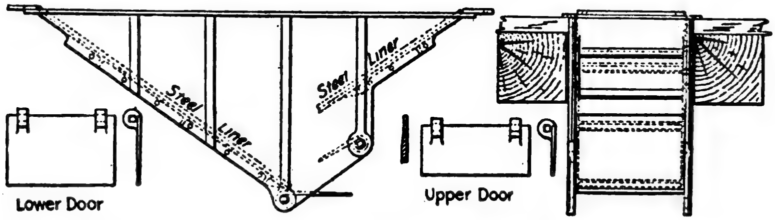


FIG. 131.—CAST-IRON CHUTE USED IN CALUMET & HECLA ROCK HOUSES.

tracks the gates are operated by a system of levers from a small house or platform on the outside of the car tunnel.

Feed Gate for Coarse Ore.—In Fig. 132 are shown the details of the gates used on the hoppers that feed coarse ore to No. 6 Gates gyratory

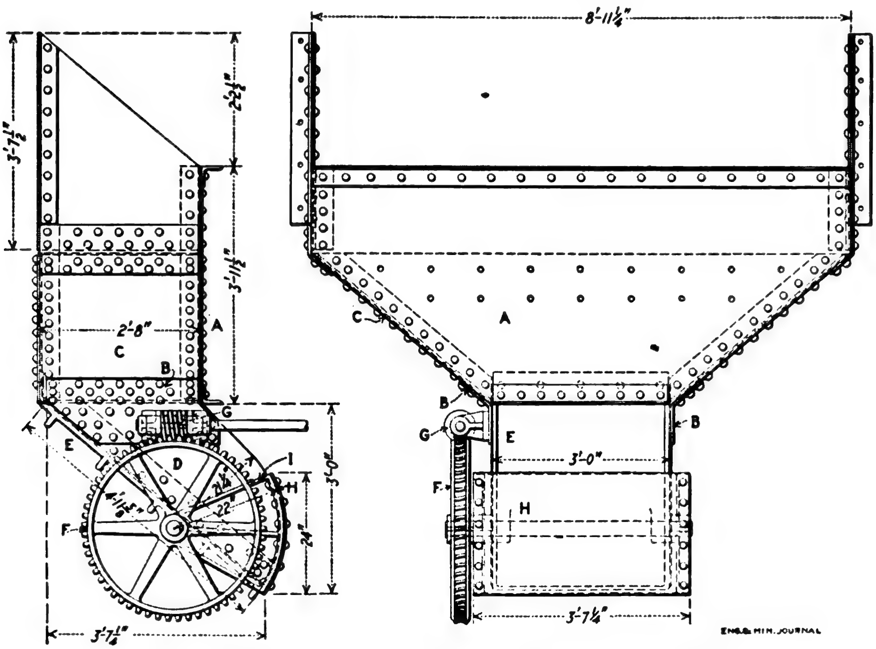


FIG. 132.—GEARED ROLLER FEEDER FOR COARSE ORE.

crushers at the mill of the St. Louis Smelting & Refining Co., in the southeastern Missouri lead district. The ore going to this bin is run-of-mine ore that has not passed over a grizzly. Consequently there are many boulders in the ore which no doubt would give considerable trouble from

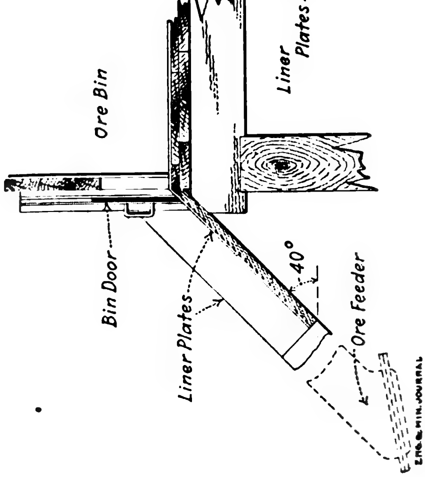


FIG. 133.—WOODEN ORE-BIN CHUTE.

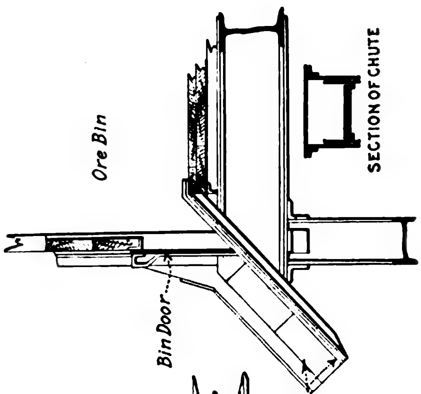


FIG. 134.—STEEL CHUTE.

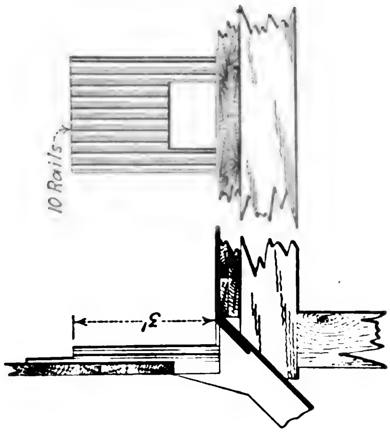


FIG. 135.—PROTECTING ORE BIN AGAINST WEAR AT DOORS.

runs in case a gate closing from above was used on the hoppers. The gate is of the bottom-closing arc type. It is attached to the bottom of the feed chute coming from the hopper that is built up of front and back plates tied to side plates *C*, by angles. To this hopper the feed chute is tied by $\frac{7}{16}$-in. plates *B*. The bottom of the chute is a ribbed casting *E*, bosses on the back of which serving as bearings for the gate shaft. To this shaft are keyed two gate straps *I*, to which the gate plate *H*, is bolted. On the overhanging end of the gate shaft is keyed the toothed wheel *F*, which is operated by wormwheel *G*, carried in bearings bolted to the sides of the feed chute and through the splice plates *E*. On the end of the wormwheel shaft is a handwheel by which the attendant raises or lowers the gate. Owing to the leverage he can turn the wheel easily, but the gate moves slowly. An air piston could be arranged to operate the gate if desired. The most interesting feature is that the gate closes from below so that the ore is stopped by damming it back at the angle of repose. Owing to the angle at which the feed chute is set, the gate does not have to be lowered all the way down and the ore feeds down to the crusher on a bed that is dammed back on the cast-iron bottom plate. In this way wear is greatly reduced. An angle iron is riveted to the front lip of the hopper in order to reinforce it and keep it from being bent out of shape by the boulders which often roll down with a considerable momentum. This feed gate works well. Its only drawback is that it cannot rapidly be closed in its present form, but that is easily changed if desired. Its great advantage is that it can be closed at all times without any possibility of becoming blocked by boulders.

Ore Bin Chutes in Rand Mills.—In most of the large gold mills on the Rand the ore is fed from the bins to the mortar-box through a gate provided with a chute generally set at an angle of 40 deg. from the horizontal. The door proper at the ore bin is merely a sliding gate lifted by rack and pinion, or by hand, and the chute may be of wood, as shown in Fig. 133 or of steel, as shown in Fig. 134. In either case, states C. O. Schmitt in "A Textbook of Rand Metallurgical Practice, Vol. II," the chute is lined with steel plates, easily renewable, and is provided with a removable cover, unless it is left open altogether. Considerable wear results from the flow of the ore through the opening in the bin, and the bin lining is, therefore, protected by steel plates or rails all around the opening as shown in Fig. 135. The control of the amount of ore fed to the stamp battery is not effected by the bin door, the latter being generally set by hand to give somewhat more than the quantity of feed required, while the actual regulation is effected by an automatic ore feeder.

Crib Ore Bin.—The ore bin employed for a tailing bin by the Mount Summit Ore Corporation, of Garrison, N. Y., is constructed on unusual lines. Except for a few pieces, it is built up of 1 × 6-in. material only.

Typically, 2 × 6-in. stuff would be used, and is so used in other bins of the same property. The bin is 16 × 16 × 17 ft. outside, above the foundations. These latter are of concrete, three in number, 1 ft. thick. The bin floor laid on these is of the 1 × 16-in. material set on edge and spiked together, covered with a ½-in. layer of concrete. The sides are built up of the timber laid on its side, two planks together, lapping alternately at the corners. The successive layers are spiked down to those below. The sides are plumbed up as they progress just as a mason plumbs a vertical wall. Two ¾-in. tie rods in each direction through the bin are carried by vertical 2 × 8-in. by 12-ft. pieces on the sides. The bin is covered with a sloping galvanized roof. Two bin gates of the vertically sliding type

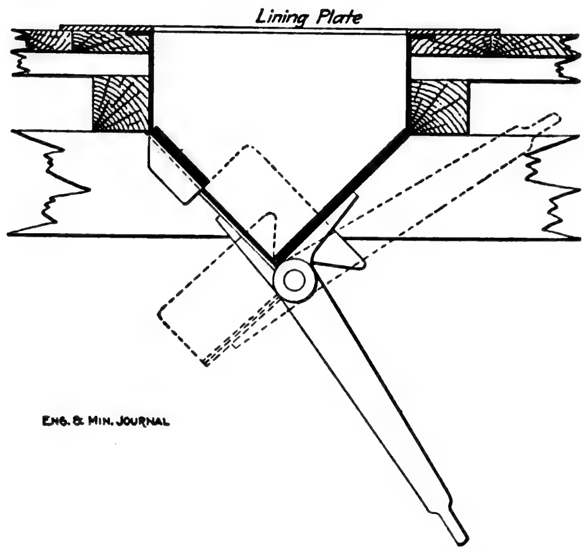


FIG. 136.—BOTTOM DISCHARGE DOOR FOR ORE BIN.

moved by a lever are set in two sides. They are satisfactory on the fine, dry, smooth-running material which they are required to handle. The guides are made of a 3-in. angle bolted to the side of the bin and a 1-in. angle bolted to this so that the gate slides behind one flange of the latter. It is expected to keep the bin filled with tailing in the lower portion and draw it off as required down to the level of the chute bottoms. An exactly similar chute set in the bottom of the bin in a horizontal position permits the bin to be forked if desired. The bin offers the advantage of rapid erection by relatively unskilled mechanics. No framing of timbers is required. It is not even necessary to cut the material to length, as the corners can be trimmed with a saw when erected. Rough hemlock is the material used in this case.

Discharge Door for Flat Bottom Bin.—The accompanying illustration, Fig. 136, shows the type of discharge door used on the flat-bottom bins in the rock houses and mills on the Rand, and which is described in "A Textbook of Rand Metallurgical Practice, Vol. II." The door is designed not only to control the discharge, but is so made with side pieces that the ore is guided in a narrow stream, so as to fall directly into the car beneath the opening without any possibility of spilling by lateral spreading out of the stream. The shape of the side pieces is such that there is no opportunity for pieces of ore to fall away from the door at any position of the chute-like door between the positions of fully open and tightly closed. The construction of the gate is clearly shown in the illustration.

Storage and Handling of Concentrates

Doe Run System of Handling Concentrates.—At the new mill of the Doe Run Lead Co. near Rivermine, Mo., the concentrates from the different parts of the mill come to a common bin where they are dewatered. The jig concentrates flow into the bin by gravity while some of the table concentrates are raised by a Frenier pump to the bin. This bin has a series of pyramidal compartments in the bottom, provided with 4-in. diameter openings through which the concentrates fall into a bottom trough along which they are moved by a scraper conveyor, up an incline at one end and discharged over the lip in a sufficiently dry condition so that they may be taken by a flat belt conveyor to the gondola cars for shipment to the smeltery of the St. Joseph Lead Co. at Herculaneum, 37 miles away. During the time that they are dragged up this incline at the end of the concentrates bin they are drained so that they do not contain more than 10% moisture upon dropping to the belt conveyor.

The bin is made of concrete laid in two layers with a coating of asphalt between, so as to make the concrete imprevious to water, and is of a rectangular section throughout, with the pyramidal bottom laid on top of a well-packed filling of cinders. Reference to Fig. 137 will give a clear idea of the design. These pyramidal walls are 7 in. thick and are reinforced with No. 12 gage, 1½-in. mesh, expanded metal. The V-shaped mass of concrete between pyramids is reinforced by three ⅞-in. corrugated bars. Also in reinforcing the horizontal bottom from which the pyramids spring, ½-in. corrugated bars are used. These are put in at 4-in. centers and are strung through two 9-in., 13¼-lb. channels that form the sides of the trough in which the scraper conveyors work. Alternate bars are turned up to take the shearing stress.

In order to enable the contents of any pyramid to be re-treated, if desired, 1½-in. pipes were buried in the concrete, but these have never been used. The outer wall that helps carry the tank is cut into a series

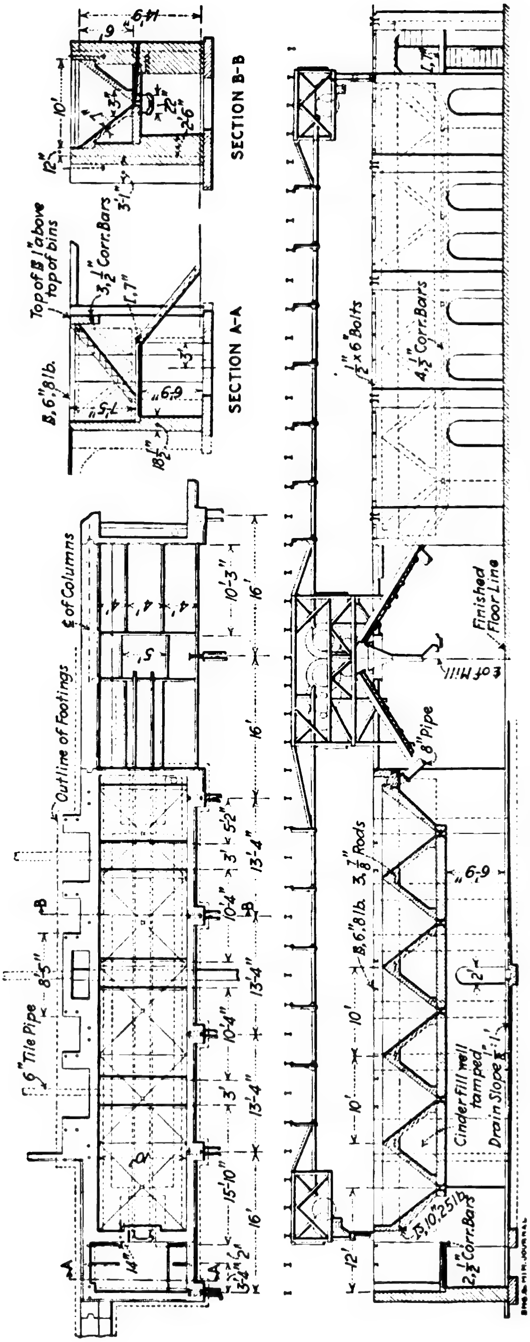


FIG. 137.—CONCENTRATE-HANDLING BINS.

of arches, reinforced by four ½-in. corrugated bars, so as to help take the tensional stress over the arches; I-beams are used to reinforce the end walls. A 1:2:4 concrete mixture was used, the rock in which had been broken to pass a 1½-in. ring. When the bin was finished trouble was experienced from leakage around the corrugated bar reinforcements that supported the trough. Some of the concrete had to be chipped out and a calking of lead wool put in around the bolts. This stopped the leaks and no further trouble was experienced. The troughs in which the scrapers travel are made by bolting a cast-iron bottom to the channels. In order to allow the bolt to be put in easily and the assembling of the trough to be accomplished without any difficulty, the bolts go through slots ¾ in. wide and 1½ in. long, in the cast-iron bottom. These holes are placed at 6-in. centers along the trough.

The tracks for the rollers of the scraper coveyors consists of 2 × ⅜-in. wrought-iron straps, carried by 3 × 4 × ⅜-in. angles riveted to the channels. Top guides were put in, so as to prevent the conveyor from lifting off the track at the turn. These are 2 × ½-in. wrought-iron bars carried by clip fastenings cut from 3 × 3-in. angles that are riveted about 24 in. apart to the channels forming the sides of the trough. In assembling the trough, rubber gaskets $\frac{1}{16}$ in. thick were put in at all joints to make them water-tight.

The conveyor wheels are 3 in. in diameter and are connected together by links that carry the scraper blades. These scraper blades are perforated so as to allow the concentrates to drain better. In order to prevent breakage resulting from the chain on one side being worn more than that on the other, the scraper blades are carried from the chain links by swivel connections so that one chain can work ahead of the other without throwing any bending or shearing stress on the scraper carriers.

The chain returns on an overhead track after passing around the sprockets at the end. In order to catch the drip from the returning conveyor, catch pans are carried along under the return tracks. These run down at frequent intervals to the level of the concentrates tanks so as to return the dripping. Walks over the concentrate bins are carried across where the catch pans are high up, next the tracks. The conveyors are run at a speed of about 12 ft. per min. and it takes a total of 5 hp. to run them. As the pull on the conveyor is at times between 3000 and 4000 lb. a belt conveyor is out of the question for such service. The travel of the conveyor under water and in water laden with sulphides is hard on the bearings, while there is considerable wear on the scrapers. There are two of these conveyors, one for each section of the mill and each conveyor handles about 100 to 150 tons of concentrates per day. The strains at the sprockets also come on the rollers and this is the chief cause of wear. Other than occasional casual inspection these concentrates dewaterers

and conveyors require no attention. The cost of operating them is 3c. per ton of concentrates delivered to the belt conveyor, 2c. of this cost being for repairs and maintenance.

The chief wear on this conveyor is caused by grit getting into the bearings from the rollers and the pressure and turning when on the sprockets. A conveyor lasts about 18 months. In operating the conveyor a close watch has to be kept for broken links and as soon as one is found, the feed is turned to the other concentrate bin while the injured conveyor is immediately repaired, for in case the chain should break in two, some of the bottom plates would have to be taken off to get the conveyor into the launder again.

Handling Concentrates at Small Mills.—While it is probably best practice to convey concentrates to the bins by means of water, this is not always possible. In such cases a wheelbarrow is resorted to at small mills to get the concentrates from the machines to the bins or the drying room. A method that is to be preferred to the latter, is the one employed at the Montana-Tonopah mill, at Tonopah, Nev., and at the Liberty Bell mill, at Telluride, Colo., where a tramway bucket suspended from an overhead track is used. This track is so hung that the top of the bucket is only 1½ or 2 ft. from the floor and about 3 ft. from the machines, so that the concentrates can be shoveled into it easily from the boxes.

The Handling of Wet Concentrates (By M. J. Elsing).—At Cananea, Mex., the wet concentrates from the Cananea Consolidated mill are loaded into small, narrow-gage railroad cars and transported to the main ore bins. From these bins they are fed to a belt conveyor which passes through the sampling works to the mixing beds. A simple ingenious device utilizing compressed air has been introduced which greatly reduces the cost and the labor of handling this wet material. From the jarring and the settling received in the railroad cars the wet concentrates become so firmly packed that even after the bottom of the car has been released, the concentrates will not fall from the car. Formerly it was necessary for two men to do considerable barring in order to dislodge the concentrates and empty the car. The device consists of a 1¼-in. pipe, approximately 9 ft. long, with a short piece of pipe attached, which is pointed at one end with a hole in it about ¼ in. in diameter. The short piece is attached with an ordinary pipe coupling. At the other end of the long pipe there is a valve attached and an ordinary air hose. This is then connected with an air main under a pressure of from 80 to 90 lb. A Mexican stands on each end of the railroad car, thrusting the pointed end of the pipe well toward the bottom of the loaded car, and turns on the air. The result is that with a small amount of barring the car is quickly unloaded. The concentrates are literally blown out. Formerly it took from one to one and one-half hours to unload a train

of six or seven cars. It is now done in about 15 to 20 min. The device is again used in feeding the concentrates from the steel bins to the belt conveyor. A steady feed is obtained with the expenditure of only a small amount of air and labor.

Special Feeding Arrangements

A Grizzly Crusher Feeder (By S. A. Worcester).—The grizzly crusher-feeder illustrated by the accompanying drawings is used in a 60-stamp mill in which there are three ore bins, each supplying 20 stamps. In a former arrangement of the crusher floor there had been three crushers, one over each ore bin, and these were operated one at a time. The ore was delivered at the mill by a Bleichert tramway on which one man was employed in shifting buckets at the terminal and another in tramming

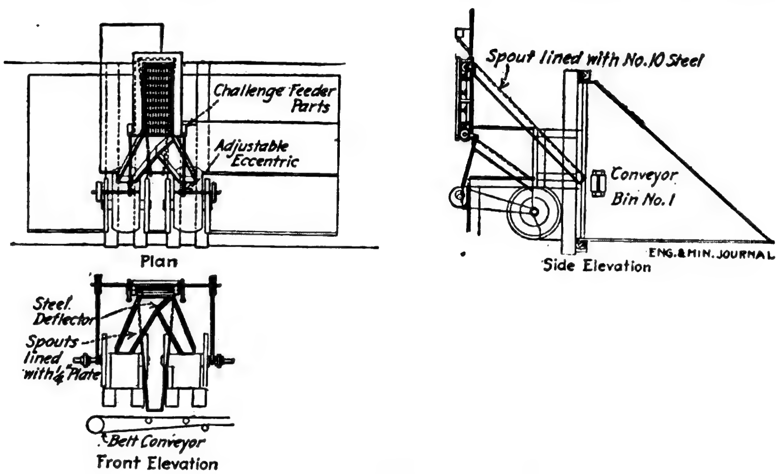


FIG. 138.—CRUSHERS WITH GRIZZLY FEEDER.

the ore from the terminal to the crushers. By removing the crusher over the ore bin farthest from the terminal and placing it at the side of the crusher over the nearest bin it is now possible to dispense with the trammer and use this crusher as a reserve. This enables the terminal operator to attend to the feeder and concentrates all the crushing at one point.

As shown in Fig. 138, the ore from the crusher and the undersize from the grizzly drop directly down to the conveyor, which delivers by trippers or by end discharge to either of the farther bins. For delivering to the bin below the crushers, swinging deflectors, which are not shown in the illustration, are used. The grizzly feeder, Fig. 139, eliminating the shovelman, operates as follows: The terminal operator dumps the tram-

way bucket of a capacity of 800 lb. of ore on the horizontal grizzly, from which the rejections are slowly and positively raked over the end and into the spout leading into the crusher. The raking arrangement is similar

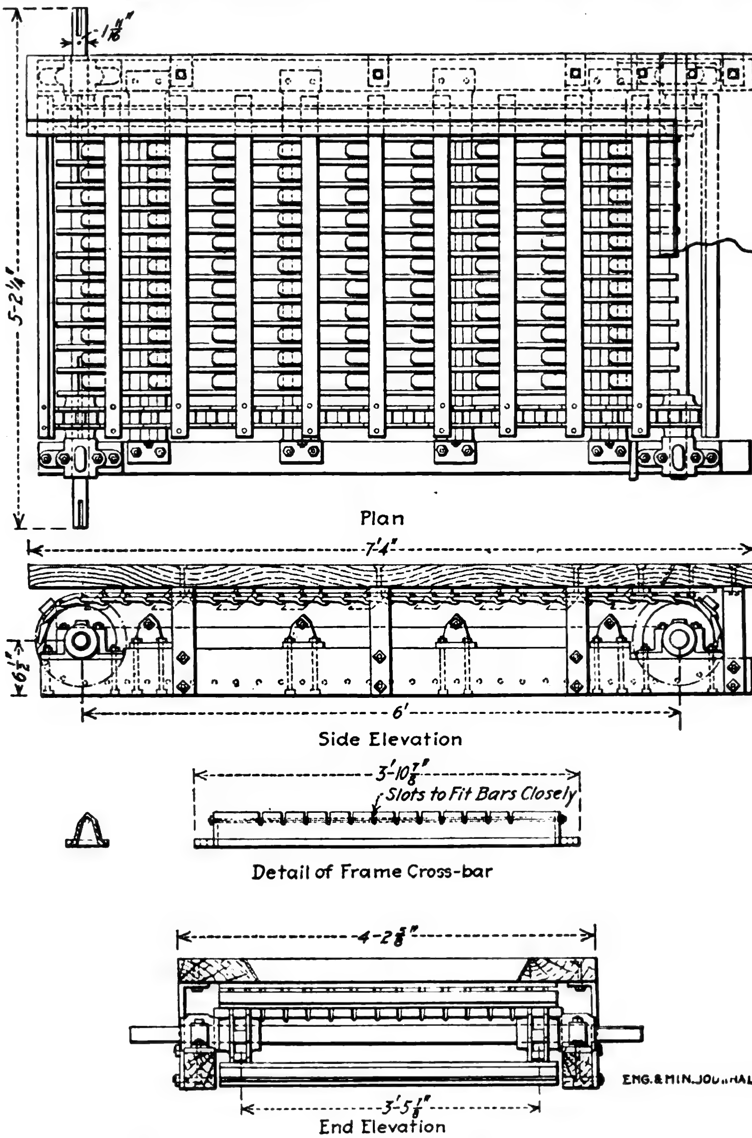


FIG. 139.—ENLARGEMENT OF THE GRIZZLY FEEDER.

to that of the chain-grate stoker used in boiler furnaces, and consists of two chains running over sprocket wheels and carrying a number of cross-bars provided with lifting fingers which travel below the upper

surface of the grizzly and lift any ore which tends to hang between the bars.

A deflector can be swung to direct the ore to the crusher which is in operation. The driving shaft of the feeder is given its slow, intermittent movement by a Challenge feeder friction device driven from an eccentric on a small countershaft driven from the crusher. The fines from the grizzly drop down the slope beneath to the conveyor or to the bin below. The rate of feed is regulated by adjusting the throw of the eccentrics. The sprocket chains are protected from dirt by a covering of planks. By the exercise of a little care in distributing the ore on the grizzly, the operator can easily avoid choking the crusher. The tapering section of the grizzly bars prevents any material from wedging below the top surface of the bars and the lifting fingers raise all material that tends to hang. The grizzly bars are firmly supported by substantial frame cross-bars to which they are secured by rods passing through the grizzly bars.

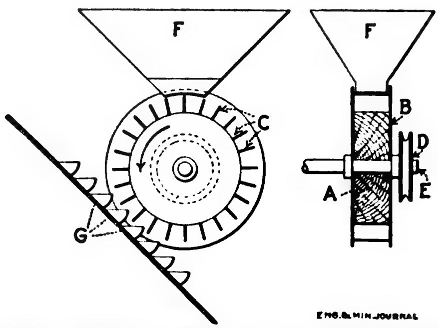


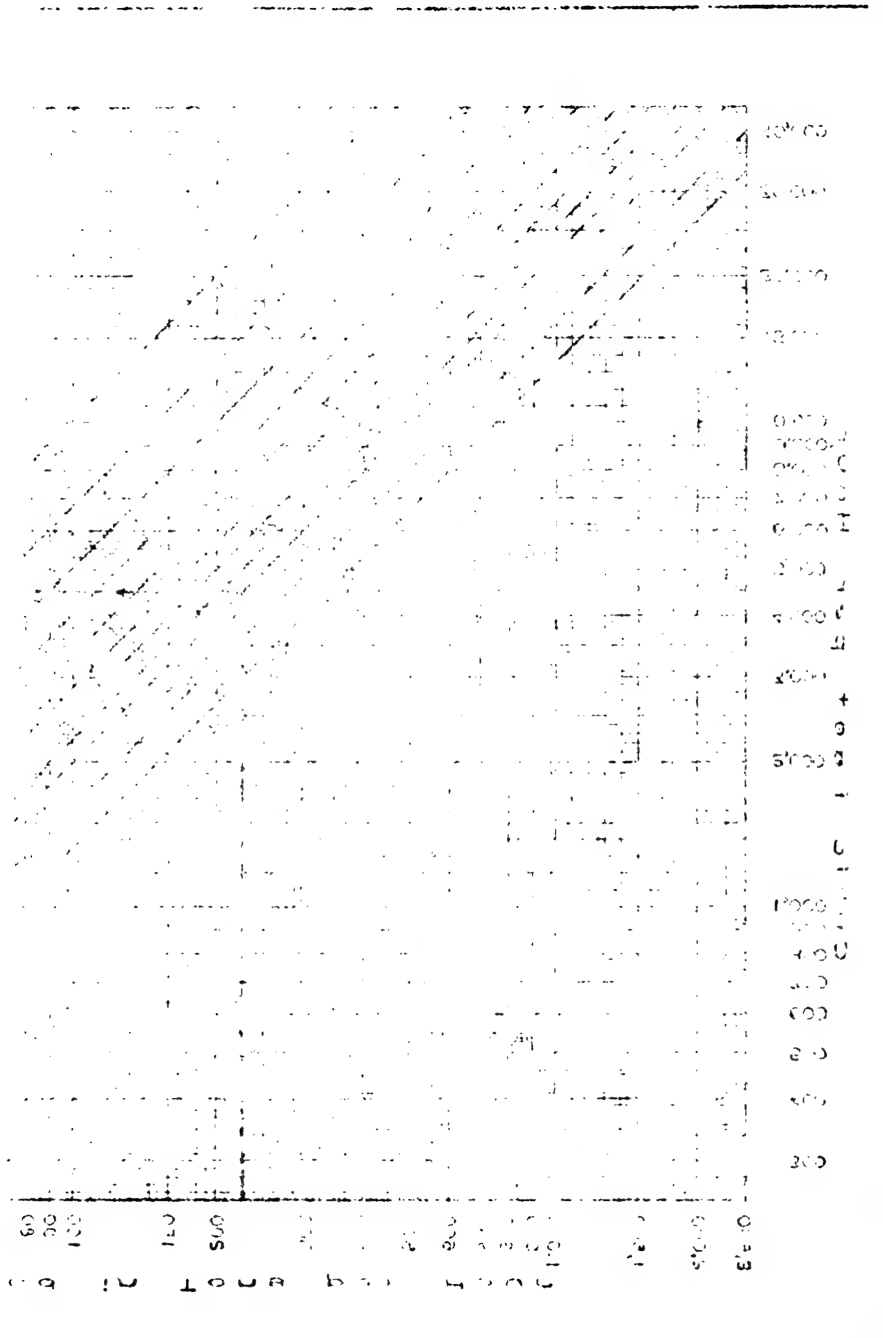
FIG. 140.—SPECIAL FEEDER FOR FINE DRY MATERIAL.

The eccentrics, shafts, pulleys and pillow blocks were all taken from abandoned jigs, and the Challenge feeder parts were likewise on hand.

A Feeder for Fine Material (By Herbert A. Megraw).—A problem in ore feeding presented itself to me not long ago in connection with the treatment of a large accumulation of dump material. This material was ore rejected in old smelting operations in Mexico, and consisted of particles varying in size from $\frac{1}{4}$ in. to -200 mesh. It was largely composed of iron oxides, which, with the proportion of slime contained, gave a decided tendency toward packing, even in the dry state. The material was to be cyanided and as it lay on a level lower than that decided upon for the grinding machinery, it had to be raised by means of a belt-and-bucket elevator. An important detail was the rate at which the material should be raised, because all the machinery following was automatic and it was essential that the ore should come up in a constant, even stream. The first feeder tried for delivering the material in the desired way to the

belt elevator, was a screw, but the fine material packed between the screw and its casing and required so much power that the device had to be discarded. Various forms of pushers were then tried but without a great degree of success. The design shown in Fig. 140 proved to be most satisfactory. It consists of a wooden wheel *A* with a sheet-metal disk *B* on each side of it, mounted on a shaft *E* which is moved by the pulley *D*. The channel formed by the wooden wheel and the sheet-iron disks is divided into a number of pockets by the divisions *C*. These divisions do not reach to the circumference of the metal disks but stop about $\frac{1}{2}$ in. short of it. This is in order to allow the small end of a hopper *F* to fit into the pockets and between the metal disks. By revolving the wheel in the direction indicated by the arrow, a constant supply is cut out of the hopper by the pockets, and delivered in a uniform stream to the buckets *G* of the elevator. The hopper may be of any convenient size and the other dimensions of the device will naturally vary as the requirements. By regulating the size of the pockets and the revolutions of the wheel, a constant uniform stream of any desired quantity may be delivered to subsequent machinery.

Traveling Belt Ore Feeder.—At the Boston Consolidated mill there was installed a feeder of the traveling-belt type, somewhat similar to those in use at coal bunkers of a few mines in Pennsylvania. When the mill was being designed, it was the intention to mine the ore by means of steam shovels. On that account, it was anticipated that there would be numerous boulders in the ore coming to the mill. To handle such a feed either a feeder of the traveling-belt type or one of the reciprocating-pan design can be used. The traveling-belt type was selected and this feeder has given excellent results after two years of service and has required practically no repairs. It was the design of A. J. Bettles. The chute openings in the sides of the bins are stopped with three inclined horizontal slats as the openings are 4 ft. high. In case of boulders or freezing causes the blocking of the chutes, the ore can be started again by punching with a bar through the stoke or punch holes. The flow of ore is stopped by the traveling-steel belt as the surface of repose of the ore intersects that of the belt. The belt is 30 in. wide and is driven by a ratchet wheel operated by a pawl from an eccentric shaft. The speed of the belt travel is adjustable by means of the eccentric arm from $1\frac{1}{4}$ in. to 6 in. per revolution of the eccentric shaft which is driven by means of a set of gear wheels run by a rope drive from the motor operating the gyratory crushers and the belt conveyor. The feeder is thrown out of gear by raising the pawl that operates the ratchet wheel. The ore from the feeder belt falls into a steel chute that has an inclination of 30 deg. in the direction of the travel of the conveyor belt so as to reduce the impact of the ore falling on the conveyor.



BELT CONVEYORS

General Notes

Capacity and Speed of Belt Conveyor.—The accompanying chart, Fig. 141, shows, almost at a glance, practically all data necessary in determining the speed, capacity and size of belt required for handling material, both sized and unsized, in lumps varying from 1½ to 14 in. in diameter, according to the practice of the Robins Conveying Belt Company.

In the chart the vertical lines represent cubic feet of material handled per hour, as read from left to right, at the top. The horizontal lines, reading up, at the left of the chart, represent the weight of material handled in tons per hour.

Crossing the chart are two series of diagonal lines, the upper representing material of different density, varying from 20 to 150 lb. per cu. ft. The lower series of diagonals represents the required width of belt for any given capacity and speed.

On the right of the chart, reading up, is a scale giving the speed of the belt, for different widths and capacities. A close inspection of this portion of the chart shows that, for the same capacity, the speed of the belt varies inversely as the square of the width; or, for the same speed, the capacity of a belt varies with the square of its width.

For engineers who desire a formula, this relation of capacity (c), speed (v) and width (w) of conveyor belts is expressed as follows:

$$\frac{c_1}{c_2}=\frac{v_1}{v_2}\left(\frac{w_1}{w_2}\right)^2$$

In comparing two conveyor belts operating under like conditions and handling the same material, the tonnage ratio is always equal to the product of the speed ratio and the square of the width ratio. For example, suppose a 16-in. belt, running at a speed of 150 ft. per min., delivers 60 tons of a certain material per hour, and it is desired to find the tonnage of a 40-in. belt, running at a speed of 80 ft. per min. Call the required capacity of the 40-in. belt x; then, applying the above formula,

$$\frac{x}{60}=\frac{80}{150}\left(\frac{40}{16}\right)^2=\frac{8}{15}\left(\frac{5}{2}\right)^2$$

$$x=\frac{60\times 8\times 25}{15\times 4}=200 \textit{ tons per hour.}$$

The heavy line crossing the chart shows that a 24-in. belt would have to be run at a speed of 240 ft. per min., in order to handle a tonnage of

225 tons per hr. of material weighing 100 lb. per cu. ft. The two scales at the lower left-hand corner of the chart show what width of belt is preferable for handling different sizes of material, the inner scale being for lumps of a uniform size and the outer one for mixed material, the scale reading indicating the maximum size of lump. The heavy line just referred to is for a belt handling mixed material when the lumps do not exceed 8 in. With finer material when the lumps do not exceed, say 5 in., there would be required in the case cited above, a 20-in. belt running at a speed of 350 ft. per min.; or, for 14-in. lumps, a 30-in. belt, running at a speed of 155 ft. per minute.

Cubic feet per hour can be converted into tons per hour, for material of any given density (weight per cubic foot) by entering the chart at the top and following down the vertical line corresponding to the given cubical capacity, to its intersection with the diagonal corresponding to the given density or weight of material. From this point follow the horizontal line to the tonnage scale on the left. By reversing this process the cubic capacity corresponding to any given tonnage and material may be found.

In all cases the lower diagonals for different widths of belts are terminated at the horizontal line marking the maximum desirable speed for the belt. The speed of a belt may be augmented to carry a greater tonnage, the capacity varying in the same ratio as speed, provided the maximum speed is not exceeded.

The Abuse of Conveyor Belts (By John J. Ridgway).—It is an unfortunate fact that the purchaser or user of newly introduced labor-saving devices has to depend on the producer for such information as is necessary to determine the advisability of its introduction and purchase. If the effort were always to arrive at the facts, unprejudiced by the impress of the almighty dollar, this information might be relied upon, but unfortunately deficiencies are either evaded or avoided and when it is too late the purchaser learns to his sorrow where the trouble lies.

The engineer in charge of the construction of new plants or the renovation of old ones has often an uncertain plan for the introduction of something initially undecided upon, and leaves a space for the installation of the belt conveyor so circumscribed as to occasion unavoidable wear and tear, and the purveyor, thinking more of the immediate profit than of the ultimate effect on his reputation and the reputation of the article he purveys, concedes what he knows is wrong, to the detriment of all concerned, purchaser, producer and the reputation of the article itself.

Those who have belt conveyors in charge are often thoughtless of the risks they run when they permit a workman, in clearing the transfer chutes, to use a shovel or hoe which might be, and is often dropped from various causes—being struck with pieces of heavy material, through the

man's foot slipping or some unusual sound like the pistol shot of electric contact broken at the switchboard—and before anything can be done, hundreds of dollars worth of damage has been occasioned, when the remedy for this is equally simple. The instrument used to clean these transfer chutes ought to be attached by chain or otherwise, so that by no possibility could it drop through and do the damage above described, and yet without prejudice to the performance of its proper function.

Conveyor belts are often left fully loaded and without the necessary motor control are started up full speed, a condition absolutely unjustifiable and yet one that often occurs. The troughing idlers are often constructed more for the immediate convenience of their individual manipulation than with the thoughtful consideration of the wear and tear of the belt which first, last and always being the large item, ought to be considered first. In my judgment, pulleys in line, when driven, have a tendency to act like a pair of shears, and this is continuous and intensified when the loads are heavier than they ought to be or the carriers are spaced too wide apart, and is a fruitful source of destruction to the belt. As a rule, in belt conveyors of any appreciable length, the belt represents at least two-thirds of the initial cost of installation, and it would hardly seem the part of wisdom to prejudice two-thirds which is subject to constant wear, for a saving in one-third which, in the nature of the material and the character of the work that it performs, under proper care and management should last almost indefinitely. Such, however, is the case.

In all other kinds of mechanical devices the construction and adaptability of the article in question is always the prime consideration in making a choice and this maintains in conveyor belts, or should, more so than in most things, because of its large initial cost and its liability to injury through misuse and abuse. The criterion of merit in an article for commercial uses is that it should represent, in service or efficiency, the greatest possible return for the amount of money expended, and right here comes in the everlasting question of last cost. I know of cases where conveyor belts have cost $12 per ft., in contrast with others that have cost $1.75 per ft., operating for the same time, under almost identical conditions and handling the same type of material. In the first case the sharp pencil was used with telling effect on the figures of first cost of the belt, and in the second case a more far-seeing judgment was exercised.

Side Tip for Conveyor Belt.—A simple and effective side tip for a 14-in. conveyor belt, devised by W. J. Nichol, Lord Nelson Co., St. Arnaud, Victoria (*Min. and Eng. Rev.*, May, 1913) is shown in Fig. 142. As at present arranged the ore from a rock-breaker is delivered by the belt at three points, two of the points being fitted up for side delivery, as in the sketches, and the other point is over the end pulley. When the side

tip is in operation the lever *A* is raised and held at an angle by a pin at *C*; when the lever is in its lowest position the ore on the belt is carried past the tip. With this arrangement the wear and tear and strain on the belt is much less than with the usual mechanical distributor, and it is inexpensive to install, as the apparatus is merely a rearrangement of one set of three troughing pulleys, two of them being used as carriers on the lever *A*, and the third one as a guide pulley running on a vertical spindle, as shown at *D*.

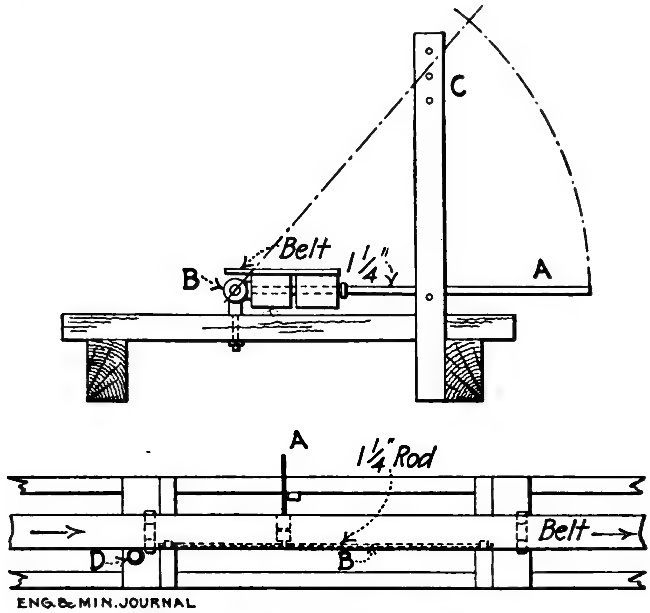


FIG. 142.—SIDE TIP CONVEYOR BELTS.

Belt Cleaners

Belt Conveyor Brush (By W. O. Borcherdt).—When handling damp or wet materials containing fines on belt conveyors the problem of removing that part of the material which does not discharge by gravity is constantly present. The most practicable method, unless a jet of water is allowable, consists in the use of the little machines with enclosed gearing furnished by the conveyor manufacturers, and with merely damp materials the rattan brushes supplied as part of these machines are perfectly satisfactory, when run at 500 or 600 r.p.m. When handling wet materials like fine jig or table tailings, however, the brushes must be set up hard against the belts, which causes the consumption of considerable power and the rapid cutting away of the bristles. The action is the same as that of the ordinary house broom. When sweeping light, dry dust from a floor the broom

may be held so that the bristles touch only at the ends, and little effort is required, but if the floor is covered with wet sand, pressure must be put on the broom, so that the bristles bend, and considerable effort is required. The consequence is that the wear, instead of coming only on the ends of the bristles, comes some distance up, the bristles are worn through, and rapidly cut off.

In place of the bristles another expedient consists in mounting on the brush shaft a four-, six-, or eight-sided block, on the faces of which flaps of rubber belting are nailed, so that when the brush is revolved, the flaps are thrown outward by centrifugal force, and sweep the belt surface. Here again, however, the rigging must be set up so that, instead of merely the edges of the flaps sweeping, an inch or two of each flap slaps against the belt and drags along it, forcing the sharper grains into the belt surface, the wearing out the flaps. My modification of this arrangement consists, as shown in Fig. 143, in mounting, by means of sloping end blocks and wooden spacers, disks of rubber or canvas belting 12 in. in diameter on the

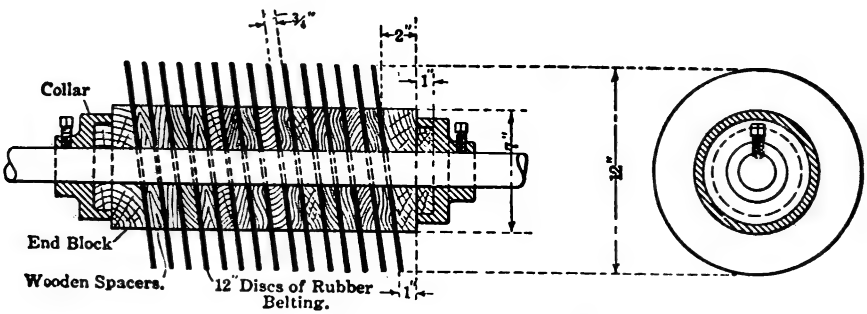


FIG. 143.—CONVEYOR BRUSH MADE OF RUBBER DISKS.

brush shaft so that they maintain a fixed angle to it. When the brush is revolved the disks stand out stiffly under the influence of centrifugal force, and a moment's consideration will show that a track which each disk makes on the conveyor belt in contact with it is a wave and that it removes the particles in its path entirely by the action of its edge and with a side-to-side shearing motion practically parallel to the shaft, instead of by throwing them directly back from whence they came. The disks are spaced closely enough so that the spaces cleared by each disk overlap, and only enough tension is needed on the take-up to maintain the edges of the belt disks in good rolling contact with the conveyor.

Direct experiment has shown that one set of disks arranged in this manner will outlast three rattan brushes under the conditions stated and the labor required to cut the disks and mount them is a small item after the blocks are once prepared. The easiest way to make the blocks is to

turn them up in the solid, bore the shaft hole, and saw them apart at the angle desired. The wear on the belt disks is entirely on the edge, they remain circular, and can be used until worn down to the wooden spacers, when the end casting is removed and new disks inserted, nailing each to the spacer ahead of it with one shingle nail, which also serves as a mark to locate the inclined center line.

An Automatic Belt Cleaner (By John J. Ridgway).—A certain amount, depending upon its character and moistness, of the material carried upon a conveyor belt, tends to cling to the surface. This clinging material

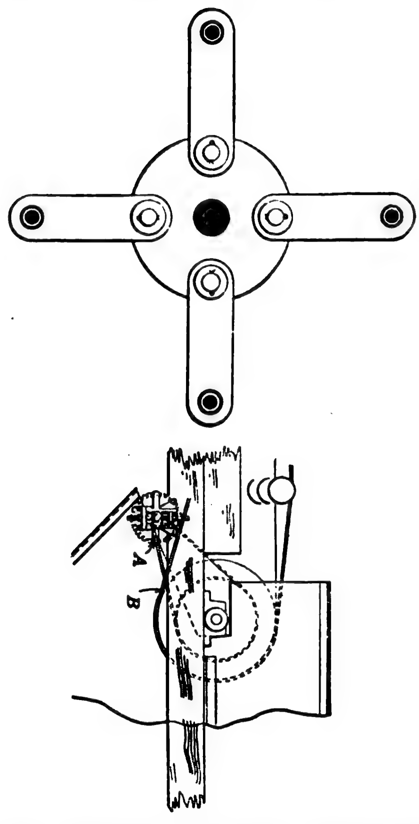


FIG. 144.—DEVICE FOR CLEANING CONVEYOR BELTS.

causes deterioration which may vary from 5 to 25% of the total wear on the belt. Various means have been tried to remove this clinging material, but all on the one principle of brushing; making the brush of fiber, bristle, etc. Not only does this type of brush or cleaner necessitate an almost constant adjustment to compensate for the wear of the brush itself, but its mechanism is complicated and if a counterweight is used and the pres-

sure made constant, then instead of becoming an aid to the longevity of the belt, it becomes a source of destruction.

In Fig. 144 is shown a type of cleaner that is effective and easily applied. The cleaner *A* is mounted on the framework carrying the end pulley of the conveyor so as to be a little below the bottom of the returning portion *B* of the belt. The cleaner consists of two disks mounted on a shaft, at a distance apart a little greater than the width of the belt. In four holes bored near the outer edge of the two disks, four arms are inserted as shown in the upper drawing in such a manner that they may swing loosely in the holes in the disk. The shaft is carried in bearings and is driven by a pulley, belt connected to a pulley on the end of the shaft of the end conveyor pulleys.

By centrifugal force, the loosely mounted arms fly outward and strike the surface of the passing belt above, effectively jarring all clinging material loose. The loosened material falls into a hopper placed just below the device. The blow may be made light or hard by adjusting the bearing screws; the amount of interference of arms and belts is not less than 32 in. and may be made as much more as desired. The beater arms, where they come in contact with the belt, being in the form of tubes, act as rollers which revolve in the direction of the belt's travel, thus eliminating objectionable features of wear on the belt which might be excessive were tubes not used.

BUCKET ELEVATORS

General Notes

Bucket Elevator Chart.—The accompanying chart, Fig. 145, shows practically all the data necessary to determine the speed, capacity and size of bucket elevators, according to the practice of the Robins Conveying Belt Co., of New York. In using the table it should be borne in mind that the required bucket capacity found in the diagram is the working capacity, and somewhat larger buckets should be used, depending on conditions. It is common practice to assume the working capacities of Salem buckets as two-thirds to three-quarters of total capacity when they are operated on a vertical belt. The heavy line gives the solution for 65 tons of coal per hour, at a speed of 175 ft. per min., and a spacing of 21 in., showing required working capacity of 740 cu. in. To find the horsepower required for driving a vertical bucket elevator, multiply tons per hour by lift in feet and divide by 500. This allows for 100% friction.

Joplin Types of Bucket Elevators.—Two types of bucket elevators are in general use throughout the mills of the Joplin district. These are known as the mill or "dirt," and tailing elevators. The former are built inside the mill, are of low lift, 28 to 30 ft., and are required to

raise the pulp from one piece of apparatus to another through which the pulp must pass in the process of concentration, because it is the practice in the district to place all mill apparatus, excepting screens and trommels, on the floor. The tailing elevators are often built outside the mill and are sometimes 60 or 80 ft. high in order that a high dump may be built up. At some of the mills, especially those at mines of unusually long life for that district, a dummy elevator rope driven from the first elevator is used to elevate still further the discharge from the tailing elevator. Both types of elevator are driven slowly, and in order that the wet material may discharge cleanly the mill elevators are built at a pitch of 1:6, while for tailing elevators a pitch of 1:8 or 1:9 is common.

A satisfactory construction of elevator for raising wet material has been evolved in the district, but it may be said that in general the pulleys used are too small. The smaller the pulleys the greater the difference between the strains on the two sides of the belt. In order to secure longer service, belts of more than the usual number of plies for elevator belts are used. It would seem to be preferable to use thinner belts made with fewer plies but of greater tensile strength because with thick belts crinkling of the inner face occurs on which the wear appears to concentrate. When the belts begin to wear dirt and water enter the holes, causing blistering of the belt and enlargement of the holes, especially when passing over the pulleys. Some millmen are now beginning to use woven belts of the balata and Scandinavian types.

The tailing-elevator buckets are from 16 to 24 in. wide, 18- or 20-in. buckets usually being used; the mill elevator buckets are 18 to 24 in. wide because they raise so much material for regrinding. The buckets are made locally of No. 10 or No. 12 sheet steel riveted together; No. 8 steel is sometimes used. It is the practice to rivet a center partition in buckets that are more than 18 in. wide as this partition reinforces the bucket and tends to prevent bending of the lip.

The tailing elevators are gear driven; the mill elevators are driven directly by belts. The elevator pulleys are from 18 to 36 in. in diameter, the smaller sizes being more common. A belt speed of 325 to 400 ft. per min. for elevators driven by a small pulley is usual, but in the best practice the speed is 260 to 300 ft. per minute.

The elevator housings are always made with a sloping front and vertical back, so that all spilled material may fall unimpeded to the boot. In order to give access to all parts of the elevators when it is necessary to make repairs, the housing is built so that there is 15 in. clearance on each side of the belt. The front of the housing at the mill floor is removable and is held in place by buttons, so also is the front section of the hood, the top of which is removable, and a ladder is provided at the side of the high tailing elevators. Below the floor line, the front of the

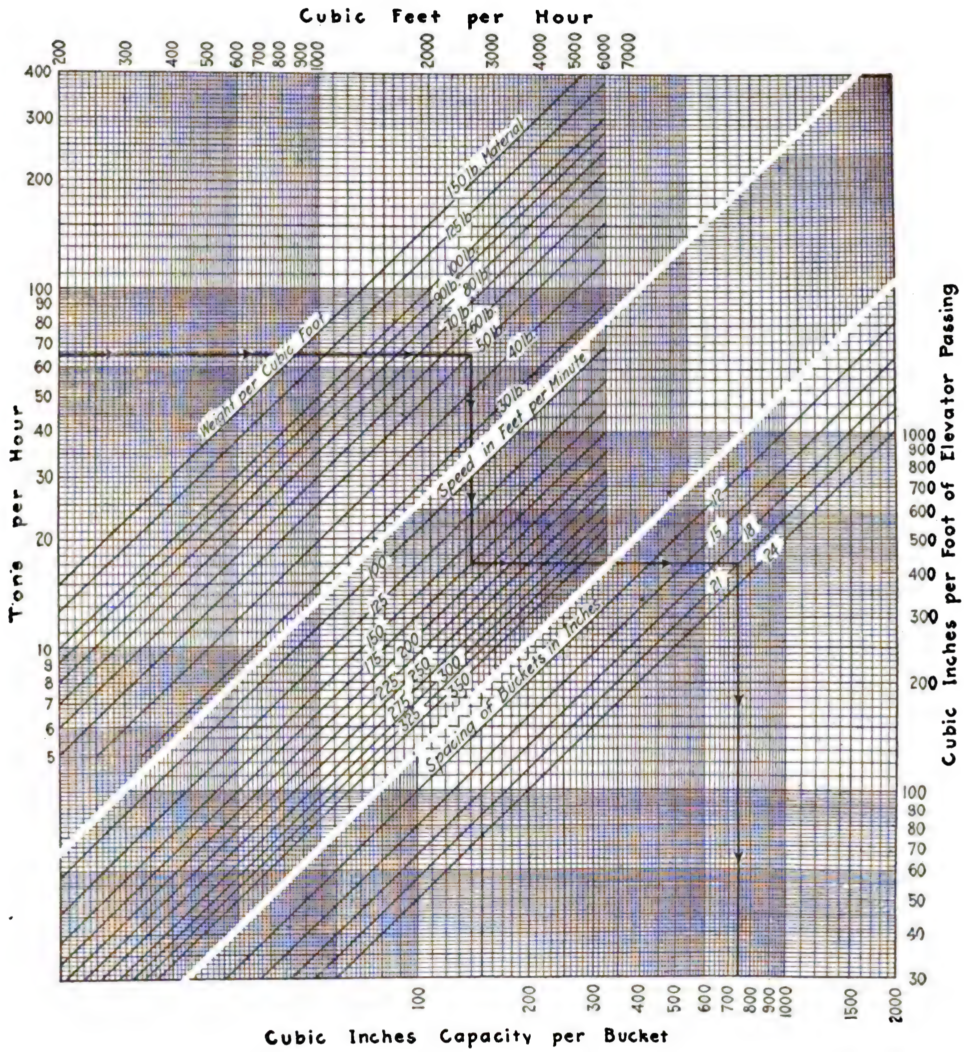


FIG.—145. CAPACITY AND SPEED CHART FOR BUCKET ELEVATORS.

To find the proper size of a bucket elevator for a given tonnage or volume of material per hour and given weight per cubic foot, enter chart at left with tons per hour and follow across to diagonal line showing weight per cubic foot, thence down to diagonal line representing the desired speed of elevator, thence across to the right until the diagonal showing desired spacing of bucket is reached. At the bottom, directly below the intersection, is found the required capacity of bucket in cubic inches. If the hourly capacity is given in cubic feet, enter at top of the chart and follow down until the desired speed line is found, thence across to spacing, etc., as before.

housing is generally made 12 to 18 in. wider than above and a ladder is put in to afford access to the boot. The clearance between the belt and the front of the housing is 12 in. At the top the housing is contracted 6 in. on each side.

A concrete boot is used in some mills, but it is the usual practice to use a simple box 12 in. deep. The feed box is put in front of the well and about 12 in. above the point where the buckets take the feed. The catch box at the top of the elevator is placed low so that the dump board can be put in under the overhang of the bucket and still allow 2 in. clearance between the lip of the bucket and the dump board. Then no matter how slowly the belt moves, the spillage at the top will run into the catch box and not fall into the housing. The top of the dump board is capped with a piece of pipe, which is quickly worn through by the tailing which then packs in the trough formed by the remnant of the pipe, making a bed of tailing to protect the edge of the dump board from wear.

The shaft of the lower pulley is carried in grooves cut in the ends of two pieces of 6 × 6-in. timber, which slide in guides nailed to each side of the elevator housing. These sliding timbers can be depressed by levers to tighten the belt. The levers are then held in place by pins. A bolt is often used to close the forked end of these sliding timbers, so that the lower pulley will not drop out should the belt break. The housing of the elevators is generally made 10 in. wider on each side below the floor line, and the take-up device is placed below and inside the housing, so that the overhang protects it from wear from the spillage falling into the boot.

The frame of the housing of the tailing elevators and some of the larger mill elevators is made of 6 × 8-in. main timbers. The boxing of 1 × 12-in. boards is nailed to the inside of the main timbers, 2 × 6-in. timbers being used in the corners to make box joints. Putting the lining inside the frame prevents wear of the frame by the spillage and also prevents the main timbers from becoming waterlogged and warping out of shape upon drying when the mill is shut down. In building the smaller mill elevators, 2 × 6-in. planks nailed into a box corner are used as the main frame timbers for the housing. In the tailing elevators 2 × 6-in. diagonal braces are used to gain greater stability and often a brace is carried back to the frame of the mill building. The highest of these elevators are steadied by $\frac{3}{16}$-in. guy wires or old hoisting cables.

The bearings of the lower elevator pulley are not lubricated, as they generally turn in the water that accumulates in the boot. The upper pulleys of some of the mill elevators are equipped with large grease cups, but in others the bearings are so inaccessible that they get little lubrica-

tion. Hot boxes in the mill elevators have been known to start fires that caused loss of the entire mill.

Chat Elevator and Loader.—A large amount of chats (tailings) in the Joplin district is sold to the railroad companies for ballast. The handling of a large tonnage of this material has evolved the construction of an elevator for loading railroad cars. The elevator is run by a 35-hp. motor which takes its power from the trolley system. The elevator consists of three belt elevators, inclined at an angle of 45 deg. and having 14-in. buckets. These three elevator belts are operated on three sets of belt pulleys, attached to the same shaft. Fig. 146 shows the general arrangement of the elevator. The entire outfit is mounted on two pairs of car wheels with 6-ft. wheel base. The lower shaft to which the pulleys are attached is 10 ft. long. At each end is a wheel *a*, 24 × 30 in. which was originally a belt pulley. To the rim of this wheel is bolted a curved piece of 3 × 3-in. angle iron, which forms a spiral, one wheel being right and the other left. These wheels assist in feeding the bucket elevator by throwing the chats to the center. As the material is elevated it falls into a chute, which in turn conveys it to the car.

The apparatus is operated on a standard-gage track. Short rails are placed in front of the machine when it is necessary to move the machine forward. This part of the work is the same as for steam shovels. The loader is moved forward or backward by means of the motor car that does the switching of the cars. Two men are employed to assist in feeding the elevator, and one man operates the motor. The elevator belt is 25 ft. long, center to center of pulley. Thirty buckets are attached to each belt. It requires 5 min. to load a 15-cu. yd. car. The loader used by the railroad company is constructed in such a way that the elevator portion may be lowered in order to transfer it where trolley wires would interfere. The hinged joint is shown in the illustration.

A similar but smaller machine is being used at the Yellow Dog mine for handling tailings that are being re-treated. The loader at the Yellow Dog mine has a friction clutch which operates a chain belt connected to the wheels, so that is can be moved forward or backward by its own motive power as in the case of the steam shovel. There is also a double winding drum, which operates a car with a tailrope system. The car has a capacity of 2½ tons, and the haul is 250 ft. Forty trips per hour can be made; the car is self-dumping, and while the car is making its trip to the bin, a hopper is being filled by the elevator so that no time is lost in loading. Three men operate this machine for which a 20-hp. motor is required.

Auxiliary for Bucket Elevator.—In order to permit short shutdowns for making repairs on a bucket elevator, an auxiliary is provided in the No. 3 mill of the Doe Run Lead Co. of southeastern Missouri. A pit

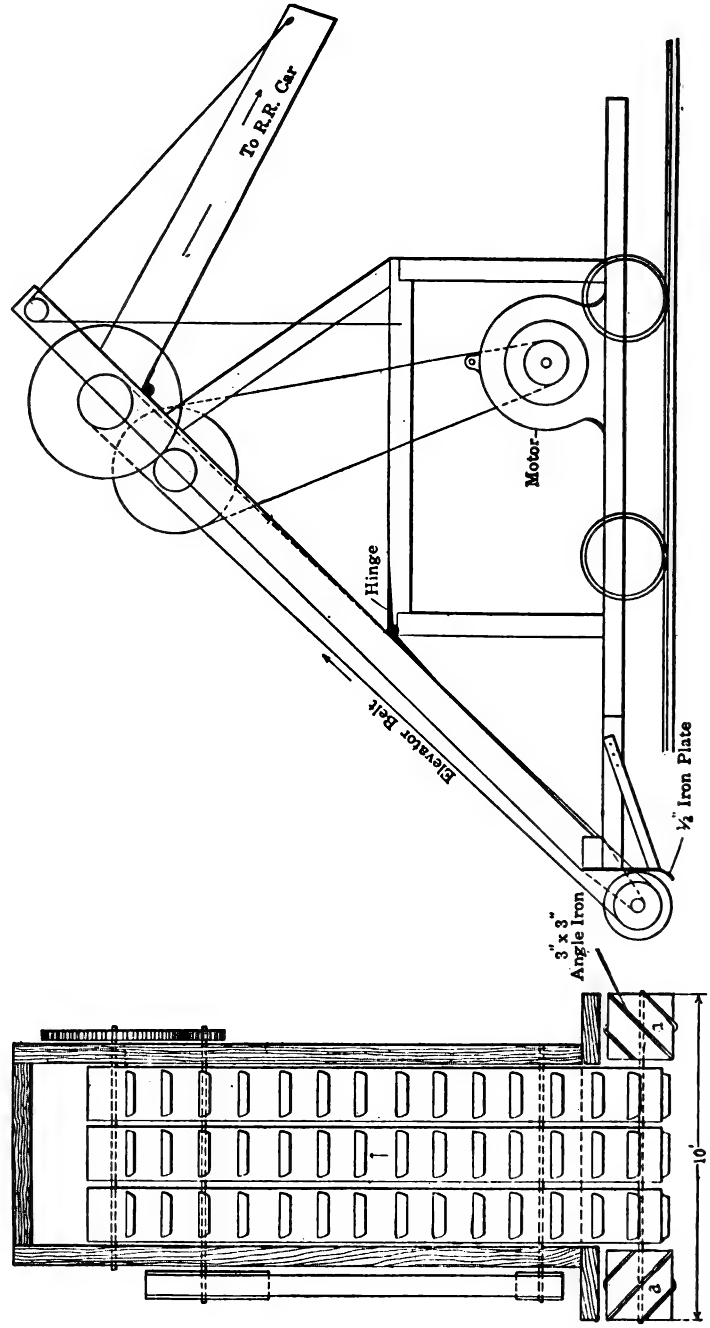


FIG. 146.—FRONT AND SIDE VIEW OF TAILINGS ELEVATOR IN THE JOPLIN DISTRICT.

was constructed next to the main elevator pit and lower than it so that when repairs are necessary the material in the latter can be run out and into the auxiliary, thus permitting a man to enter and work. Another small elevator works in this auxiliary and when the repair job is done and the main elevator in motion again the material in the lower pit is lifted once more to the main pit. The auxiliary elevator is run by a small belt from a projection on the shaft of the main bottom pulley, to the top pulley of the auxiliary.

Helping out Bucket Elevators (By Claude T. Rice).—At the Silver King Consolidated mill, at Park City, Utah, J. W. Thompson, mill superintendent, has increased the capacity of the elevators at that plant in a highly ingenious manner. The pulp that is to be handled by the elevator is first sent to a settling box and dewatered. The settled product is sent to the bucket elevator and the overflow from the settling box, which contains little that is coarser than 80 mesh, goes to a centrifugal pump which raises it to the level of the elevator discharge where it joins the underflow from the thickener. As the coarse product has all been removed from the overflow there is little wear on the centrifugal pump and it is often possible in this way not only to decrease the elevator repair bill, but to reduce the power consumed in raising the pulp, as the centrifugal pump can discharge the overflow at a somewhat lower point than is possible with the bucket elevator. Moreover, the fluid comes to the pump at a higher level than is the case with the elevator, owing to the depth of the boot. This method of aiding overloaded elevators has been used at the Silver King mill for over two years and is found to work admirably, decreasing the cost of upkeep on the elevators. The centrifugal pumps are used both to assist the elevators, and also to elevate the tailing before it is discharged. Some small holes are bored in the bottom of the buckets, the object being to let the coarser sand work through and drop on top of the load in the bucket below. In this way the coarse portion is kept from packing on the bottom. The holes must be of such size that the pulp will barely work through, the object being to keep the bottom sand in the bucket barely moving. No novelty is claimed for the use of holes in the buckets as it is a well-known practice, the novelty being the use of a centrifugal pump to handle the thin pulp and an elevator to raise the heavy portion.

Cleaning Elevator Buckets.—Elevator buckets handling damp ore containing a percentage of fine material, sometimes find difficulty in discharging the fine portion, which tends to pack in the bottom of the bucket. To avoid this difficulty, the operators at the New Reliance mill, Trojan, S. D., have devised an arrangement, described by H. C. Parmelee, in *Metallurgical and Chemical Engineering*, August, 1913. A plate of sheet iron is fitted loosely into the bottom of each bucket, and two bolts, pass-

ing through amply large holes in the bucket bottom, are riveted to it. The plate had a movement of about 1 in. In discharging the load, the plate falls and the movement and jar cleans the bucket.

Details of Design

Staggering Elevator Buckets.—The life of elevator belts may be considerably prolonged by staggering the buckets. The Continental Zinc

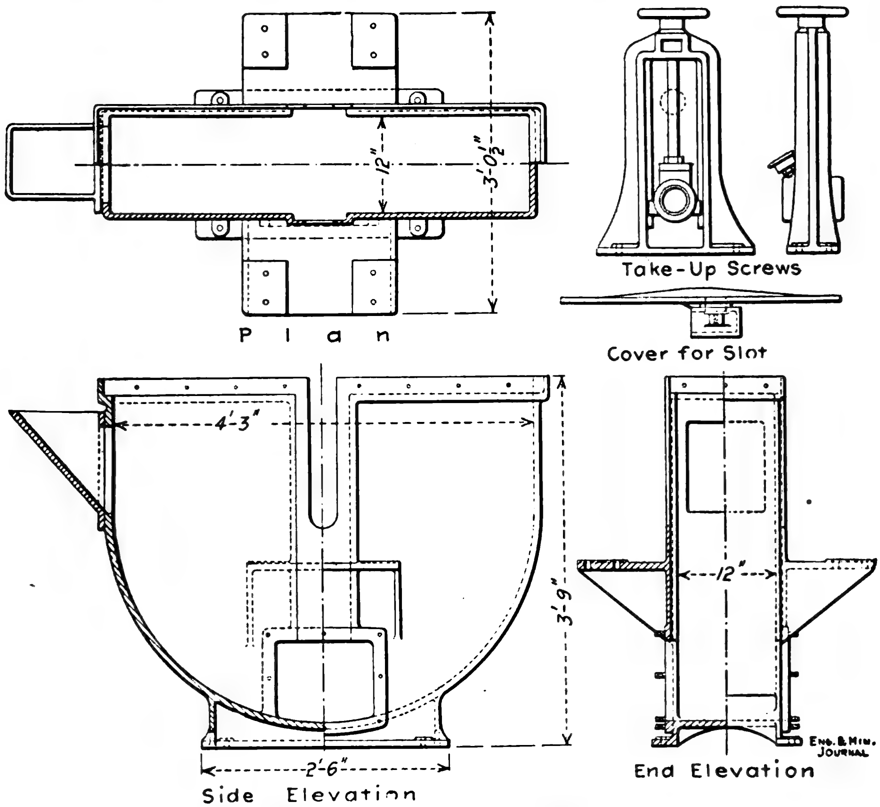


FIG. 147.—STEEL ELEVATOR BOOT WITH OUTSIDE TAKE-UP BEARINGS.

Co. of Joplin, Mo., has been using this scheme for a number of years. The belts used are 20 to 24 in. wide and in place of using buckets reaching the entire width of the belt, buckets half as long are used and placed alternately on the belt at half the interval that would be allowed for buckets of the entire width of the belt. Of course this necessitates practically the same number of holes in the belt, but they are so distributed that the line of holes does not extend entirely across the belt. Again the load is not so heavy at any one point on the belt. In case the

belt wears so much that it is necessary to shift the buckets, they are moved about half the distance to the next bucket, and at no time does a row of holes extend entirely across the belt.

A Steel Elevator Boot.—A steel elevator boot with the pulley take-up screws supported by shelves on the outer sides is used in the No. 3 mill of the Doe Run Lead Co., near Rivermine, Mo. The boot, as illustrated in Fig. 147, was designed by H. R. Wahl, mechanical engineer for the company, and is used on elevators raising dry or wet material. The boot of a wet elevator must be considerably wider than the boot of a dry elevator, although a 10-in. belt is used in each. The ends of the shaft of the lower pulley pass out of the sides of the boot to the adjustable bearings supported on the shelves through slots which are closed by slides that can be fastened in place by setscrews after the adjustment has been made. Openings are provided at the bottom of the boot through which the material can be cleaned out in case of shutdown. The feed hopper is used only on elevators raising wet material. The hopper opening can be closed with a cast-iron plate if the boot is to be used on an elevator raising dry material, as no hopper is necessary. Rubber gaskets are used in all the joints of a boot for a wet elevator. The boot is attached to the housing by four bolts so that it may be easily opened if necessary.

Bucket Elevator Catch Box.—The material discharged from the top of a bucket elevator usually drops into a catch box that leads to a chute, or when wet material is being raised, to a launder. The material is thrown or dropped rather violently on the bottom of the catch box, and the boards of which it is made are soon worn out. To prevent this wear in elevators raising wet materials, it is the practice in some of the mills in the Joplin district to put a 6-in. board or dam across the bottom of the opening of the catch box into the launder, so that a bed of pulp or tailing is retained in the bottom of the box, which effectually prevents wear from the falling material.

Belt Tighteners

Belt Tightener for Bucket Elevators (By J. Frank Haley).—Details of a belt tightener used at the mill of the Big Three Mining Co., at Bellville, in the Joplin district are shown in Fig. 148. The device is simple and can be made cheaply by unskilled labor. A piece of plank *F*, 3 × 6 in., and 6 ft. long, is cut on one end with semicircular surface. The other end is cut out to fit around the shaft of the tail pulley. This is held in a vertical position by the slot guides *M* and *N*, which are nailed to the housing of the elevator. Resting on the upper end of the piece *F* is a piece 3 × 6 in. × 8 ft. long, one end of which is bolted to the housing at *D*. At about one-third of its length from *D* it rests on the piece *F*, leaving

two-thirds of its length as a lever arm, the other end of which is shaped with a handle. The piece *C* works in a slot in the front of the elevator housing facing the jig-room floor and 3 ft. above the floor. It is held in position above by the square piece *X* and the wedge *Y* and below by the piece *A*, nailed across the front of the housing. The operation is simple. When the belt becomes loose the piece *A* is removed. Pressure is then applied at *K* and the pieces *X* and *Y* are driven in until sufficient slack is taken up in the belt. The piece *A* is then replaced, holding the entire device rigid. The shaft *E*, of the tail pulley, runs in the slots at the bottom of the piece *F*, thus doing away with babbitt bearings, which give so much trouble in belt tighteners of the screw variety.

Take-up Device for Elevator Belts.—On many of the belt elevators for raising wet pulp and tailing in the mills of the Joplin district, the screw

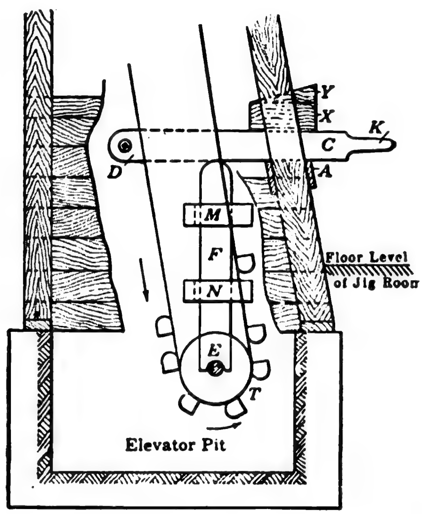


FIG. 148.—ELEVATOR-BELT TIGHTENER.

take-up device shown in Fig. 149 is used. To each side and within the lower part of the elevator housing two 2 × 6-in. guides *A*, *A* are nailed at a distance of 4 in. apart. These guides are parallel to the line connecting the centers of the upper and lower pulleys. Between each pair of guides a piece of 4 × 6-in. timber *B*, of suitable length is loosely mounted and is held from falling out of the guides by the cleats *C*. The lower end of each of these timbers is forked to carry the shaft of the lower pulley.

Slots are cut in the side of the housing and above the floor line so that a transverse 4 × 6-in. timber *D*, which is longer than the elevator housing is wide, can be extended through the elevator housing through the slots mentioned and rest between the guides and upon the upper end of each

of the timbers *B*. The slots are of such a length as to permit an up and down movement of about 18 in. for the transverse timber between the guides.

Near each end of the transverse timber *D* and outside of the elevator housing, a hole is bored through which the upper end of one of the take-up bolts *E* passes. The lower end of each of these bolts is held by a nut in the floor beams of the mill. The upper ends are threaded for a length of 18 in. or more. It will be seen that upon tightening the nuts on the upper end of each of the bolts *E*, the transverse timber *D* will be drawn down in the slots and press the timbers *B* downward in the guides, thus lowering the pulley and tightening the elevator belt. The forked ends of the timbers *B* are usually provided with a bolt *F* which prevents the pulley dropping in case the belt breaks.

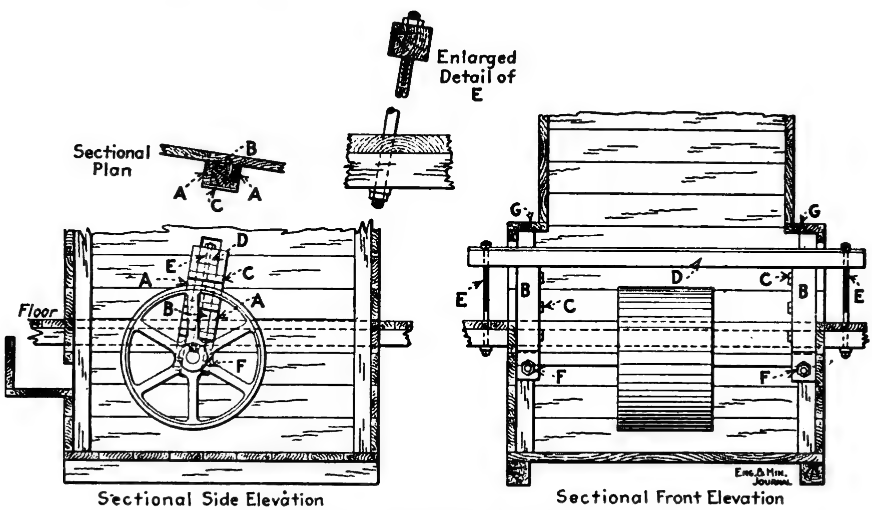


FIG. 149.—SCREW TAKE-UP DEVICE FOR LOWER ELEVATOR-PULLEY.

The elevator housing is generally made 20 in. wider in the lower than in the main part as shown at *G*, so that the overhang protects the sliding timbers and their guides from the spillage from the buckets. The device has the advantage over the lever tightener in that the adjustment part of the take-up mechanism is outside the housing.

LAUNDERS

Launder Data, Cananea Consolidated (By A. T. Tye and T. Counselman).—In Table XXVIII is given information concerning the launder system installed at the concentrating plant of the Cananea Consolidated Copper Co., Cananea, Sonora, Mex. This should prove useful as a guide to those having to install launders where it is impossible to make

TABLE XXVIII.—LAUNDER DATA, CANANEA CONSOLIDATED COPPER CO.

Material handled	Size of launder, inches depth width	Grade, in. per ft.	Lining	Ratio solids: liquids	Largest particle mm.	% Wt. on 60 mesh
Original feed to section "C"	10 × 12	3½	C.I.—11¼ in.	1:1	30.0	82.0
Bull jig tails, to coarse rolls	10 × 9½	3	C.I.— 9¼ in.	1:1.1	28.0	98.0
Roll jig tails, to fine rolls	10 × 7½	2¼	C.I.—7¼ in.	1:3	10.0	98.0
Coarse jig concentrates	10 × 7½	1⅔	C.I.—7¼ in.	1:2.9	35.0	95.0*
Undersize 2 mm. trommel to classifier	10 × 5¾	1½	C.I.—5¼ in.	1:4.4	4.0	52.0*
1st spigot of classifier to sand jig	8 × 5¾	3⅓	C.I.—5¼ in.	1:1.5	4.0	90.0
Shaking launder[1] for sand jig concentrates	6½ × 12	⅜	None—corners	1:19.0	4.0	84.0
Fine jig concentrates from shaking launder	10 × 7½	1 13/16	C.I.—7¼ in.	1:6.0	4.0	80.0
Bryan (Chile) mill discharge to drag belt	8¾ × 11½	9/16	None—corners	1:1.7	2.5	26.0
Drag belt sands to classifier distributor	10 × 7½	1⅔	C.I.—7¼ in.	1:0.9	2.5	46.0
No. 1 spigot of classifier to mud jigs	8 × 5½	1⅔	C.I.—5¼ in.	1:17.0	3.0	60.0
Table feed, tables 27 to 30	7¼ × 5½	½	None—corners	1:8.4	1.0	0.3
Table concentrates, drag belt launder[2]	9½ × 9½	⅛	1½ in. boards	1:26.0	1.0	13.6
Table concentrates, shaking launder[3]	4 × 7½	3/32	None—corners	1:7.4	0.5	35.2
Slimes, section "C," settling tanks to section "B" vanners	6½ × 9½	9/16	None	1:96.0	0.17	0.0
Slime feed to vanners, section "C"	9½ × 11½[4]	⅓	None	1:4.7	0.17	0.5
Vanner concentrates, drag belt launder[5]	10 × 25 × 16	Level	None	1:31.0	0.2	1.0
Vanner and table concentrates	6½ × 6	1 15/16	None	1:20.0	2.0	27.2
Vanner tails, section "C"	9½ × 11½	⅓	None—corners	1:7.0	0.4	0.4
Coarse tails, table and jig	9½ × 22	⅓	½-in. boards and corners[6]	1:27.0	2.0	41.6*
Coarse sand tailings to dam No. 2	12¼ × 11½	11/32	½-in. boards and corners[6]	1:21.0	2.0	19.5*
Coarse sand tailings to dam No. 1	12 × 11	9/32	½-in. boards and corners[6]	1:21.0	2.0	19.6*
Slimes to mill No. 4	9½ × 12½	¼	None—corners	1:4.4	0.1	0.0
Sands and slimes to mill No. 3	10 × 11½	¼	None—corners	1:5.0	1.5	2.6
Feed to vanners, mill No. 4	6 × 7½	⅓	None	1:4.4	0.1	0.0
Slime concentrates, mill No. 4, drag belt launder	12 × 24 × 18	Level	None[3]	....	0.1	0.2
Slime concentrates, mill No. 4, elevator to bins	7 × 7½	⅔	Glass—corners[7]	1:20.0	0.1	0.4
Slime concentrates, mill No. 3, elevator to bins	10 × 12	15/16	Concrete	1:8.7	0.2	2.8
Slime tails, mill No. 4, to Mill No. 3 settling tanks	10 × 11	7/32	None—corners	1:6.3	0.1	0.0

All cast-iron liners have 2-in. effective depth and 24-in. length, with corners rounded.

* Liable to choke. [1] Speed 160 r.p.m., actuated by heavy head motion. "Corners" are strips of wood with cross-section of 45° triangle nailed in corners of launder. [2] Speed 75 ft. per min.; old 4-in. drive belts are used, no scrapers. [3] Speed 180 r.p.m. [4] Effective width 6 in. [5] Speed 75 ft. per min. [6] Boards laid with grain across direction of flow. [7] Glass liners, ½ × 4 × 14 in.

preliminary tests. In order that comparison may be made the character of the ore handled is also given.

The country rock in which the copper ores are found is principally diorite. The remainder of the ore is obtained from deposits in limestone, quartzite and quartz porphyry. The principal minerals in the concentrating ore are pyrite, chalcopyrite, bornite, chalcocite, native copper, garnet, sphalerite, galena and oxides and carbonates of copper.

The analyses of crude ore, shown in Table XXIX, represent the typical sliming and non-sliming ores. Although so different in physical characteristics as regards slime-forming qualities, their chemical composition is remarkably similar.

TABLE XXIX.—ANALYSES OF CRUDE ORE AND TAILING AT CANANEA CONSOLIDATED CONCENTRATOR

	Copper %	Silica %	Alumina %	Iron %	Lime %	Sulphur %
Crude ore:						
Maximum slimes, rock decomposed.	2.42	51.0	14.3	10.8	1.2	10.5
Minimum slimes, rock hard....	1.50	57.0	12.7	6.2	1.0	6.6
Concentration tailing:						
Coarse tails, to No. 2 dam.....	0.72	67.6	17.5	1.8	1.2	1.4
Slimes to No. 4 mill..........	1.02	76.6	11.3	2.2	1.0	4.0

The concentrates assay approximately 6.86% Cu, 18.6% insoluble, 31.5% iron, 36.2% sulphur. The composition of the tailing after concentration is shown in one of the accompanying tables. The specific gravity of the pyrite is 4.84 and of the chalcopyrite, 4.17. The coarse sands have a specific gravity of 2.7 and the fine slimes, 2.8. Thanks are due to F. J. Strachan, superintendent of the plant, for permission to publish the above data.

Launder Data of the Washoe Concentrator.—Data concerning the launder system installed in the concentrator of the Anaconda Copper Mining Co., at Anaconda, Mont., is given in Table XXX. The grades shown are not necessarily the minimum at which the given pulps will flow, but are all safe slopes. Where extra headroom was available, it was utilized in steeper launders. There are a great many factors which effect the flow of pulp in launders which must be taken into consideration in laying out a launder system. The more important of these factors are: The sizes and specific gravities of the grains of solid matter to be transported, the shape of the grains, the sharpness of the fractures, the density of the pulp, the depth of the stream, the presence or absence of slime and the character of the launder material in contact with the pulp. All of the launders in the Anaconda mill are constructed of wood and are rectangular in cross-section. The gangue material in the ore treated at

TABLE XXX.—LAUNDER DATA OF THE WASHOE CONCENTRATOR, ANACONDA COPPER MINING CO.

Material handled	Size of launder, inches		Grade, in. per ft.	Lining	Ratio solids to water	Diameter largest particle, mm.	Depth of stream in launder, in.	% of solids on 60 mesh	Remarks
	Depth	Width							
Undersize 7/8-in. round-hole trommels	15	10	2.0	Cast iron	1:3.5	21.0	1.25	82.0	Contains slime
Undersize 5 mm. round-hole trommels	7	8½	3.0	Cast iron	1:5.2	5.0	1.25	75.0	Contains slime
Undersize 4 mm. round-hole trommels	9½	10½	1.75	Cast iron	1:5.0	4.0	1.50	50.0	Contains slime
Undersize 2½ mm. round-hole trommels	7	8½	2.5	Cast iron	1:8.2	2.5	0.50	62.0	Contains slime
Hutch product of harz jigs	7	6½	1.0	Cast iron	1:22.7	6.0	1.00	95.0	No slime
Feed to Huntington mill	7	6	1.9	Cast iron	1:3.6	2.0	1.25	81.0	No slime
3/8-in. concentrate	10	9	1.3	Cast iron	1:12.8	7.0	1.00	100.0	No slime
Evans jig concentrate	9½	9½	1.2	Cast iron	1:27.8	2.0	1.50	72.5	No slime
Coarse table feed	7½	7	1.2	No lining	1:14.2	0.3	0.50	7.0	Contains slime
Fine table feed	5½	4½	1.2	No lining	1:13.0	0.2	0.75	0.0	Contains slime
Table concentrate	11	8¼	0.75	No lining	1:15.6	0.2	0.75	0.0	No slime
Table middling	4½	5	0.75	No lining	1:15.6	0.2	0.75	0.0	No slime
Table tailing	7	7½	0.75	No lining	1:10.0	0.3	0.50	10.0	A little slime
Secondary table feed, remodeled section	7	7½	1.20	No lining	1:7.4	0.9	0.50	44.0	No slime
Fine primary table feed, remodeled section	7	7½	1.20	No lining	1:14.0	0.35	0.50	2.0	No slime
Coarse primary table feed, remodeled section	7	7½	1.20	No lining	1:4.4	0.9	0.50	46.5	No slime

Anaconda is quartz and highly altered granite. The principal minerals are chalcocite, enargite and pyrite.

Slope of Launders.—Table XXXI containing data concerning the fall of launders for different classes of material, is taken from a description of the mill at the Lucky Tiger mine, El Tigre, Sonora, Mex., by D. L. H. Forbes (*Bull.*, A. I. M. E., August, 1912). Anyone who has been called upon to design a launder system will appreciate the practical value of this sort of information.

TABLE XXXI.—LAUNDER FALLS USED IN NO. 2 MILL OF THE LUCKY TIGER MINE

Situation	Size		Dilution of pulp $S:L$	Fall, per foot
	Width	Height		
	In.	In.		In.
Stamps to classifier	8	8	1:6	⅞
Classified coarse sand to Wilfleys	6	6	1:3	⅞
Classified fine sand to Wilfleys	6	6	1:3	¾
Slime to dewatering tanks	8	8	1:20	⅛
Thickened slime to Deister tables	6	6	1:4	⅜
Sand tailings from Wilfley tables	8	8	1:5	1
Middlings from Wilfleys to pan	4	4	1:2	1
Concentrates from Wilfleys to bin	4	6	.....	1¾
Slime tailings from Deister tables	6	6	1:7	⅜
Middlings from Deisters to 4th table	4	4	1:4	⅝
Concentrates from Deisters	4	6	.....	1⅛

Large Reinforced-concrete Launder (By Claude T. Rice).—A launder of reinforced concrete is used at the Baltic mill to carry the overflow from the tailings dewaterers. As the water is clear, it does not erode the concrete. The launder is 36 in. wide and 18 in. deep, inside measurement, and is made in 16-ft. sections that are designed to carry a total load of five tons each. Its entire length is 400 ft. The joints are made so as to take care of expansion due to changes of temperature. They are filled with strips of wood ¼ in. wide, so as to be water-tight. As in time the launder is to be buried by the tailings, it is covered with 2½-in. reinforced-concrete slabs.

The sections were cast in the mill building where sand was handy. Three sizes of material were used, consisting of three parts of ⅝-in. tailings from the jigs, three parts of ordinary mill tailings, running from ¼ in. to 10 mesh in size, one part of fine sand from the regrinders, and one part of cement. Strands of old hoisting cable were used for reinforcement and the cost of the launders was only about $0.75 per ft., while an iron pipe of equivalent capacity would have cost $2.40 per foot.

Sections of the launder (Fig. 150) show the method of reinforcing. The main longitudinal reinforcement consisted of ½-in. strands of rope, obtained by unlaying pieces of old 1¼-in. hoisting cable. Along the top,

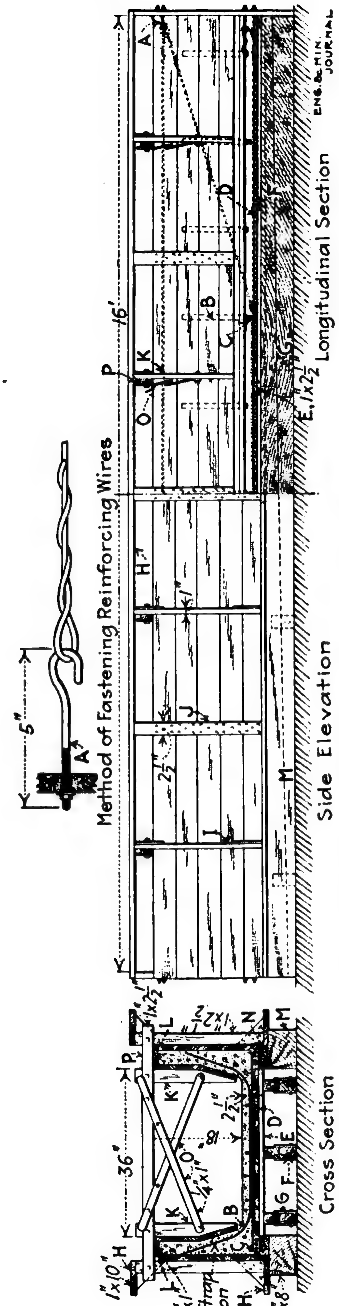


FIG. 150.—ARRANGEMENT OF WOOD FORMS FOR CONCRETE LAUNDER.

to take care of unusual tension that might result from handling during installing, a single strand was put in, while at the bottom of each side two double strands were used. These were wrapped with wire about 5 ft. from their ends, and two single strands, one for each pair, were taken up to the top of the side, while the other two were taken straight along to the end of the launder, thus caring for shearing stresses.

Before putting in the rope reinforcement, hooks *A*, Fig. 150, made of ⅜-in. rodding, threaded at one end, were fastened to the ends of each strand, the latter being heated to draw the temper, and the wires bent back and twisted around themselves so as to form loops. The threaded ends of the hooks were placed in auger holes through the ends of the forms with washers between the nuts and the wood, and the whole reinforcement tightened up so that the strands would sing when struck. In this way the stretch was taken out of the rope and with the reinforcement drawn tight, there was no danger of its being displaced during the pouring of the concrete.

To reinforce against shear between the sides and bottom of the launder, pieces of 1 × ⅛-in. strap iron *B* from the scrap pile, were put in at intervals of 18 in., running up both sides and across the bottom of the launder in continuous strips and to about every other one of these straps, to help reinforce the bottom of the launder, an extra piece *C* was added, running across the bottom only with a slight turn-up at each end for anchorage. Longitudinal reinforcement of the bottom was obtained by three single ½-in. strands above the straps. The bottom was cast 2½ in. thick and the sides 3 in. increasing to about 5½ in. near the bottom; along the bottom of the sides there was carried a ridge 4 in. wide and 1 in. deep, on which the main weight of the launder rested when set in its supports. The top slabs for covering the launder were made in 4-ft. lengths, 2½ in. thick, reinforced with three single ½-in. strands running longitudinally and short pieces of the same rope running crosswise at intervals of about 2 ft. A 1 × 1-in. jog was made in the inside top corner of the sides for the slabs to rest in.

With the forms constructed as they were, it was possible to take the side forms off in 24 hr., and so only two sets of side-forms were necessary for making the 25 sections. However, it was not thought advisable to disturb the launders in the least until they had set some time, so about five bottom forms were required. These were simple panels and were not expensive.

The forms were built up from the floor of the mill as shown in Fig. 150. The bottom panel of 1-in. planks *D*, held together by 1 × 2½-in. nailing strips *E*, about 3 ft. apart, rested upon three wedges *F*, 16 ft. long, made by ripping diagonally a 2 × 10-in. hardwood plank. To keep the top wedge from slipping sidewise off the under one, three or four pairs of

vertical cleats *G* were nailed to the sides of the under wedge to act as guides. To allow of these wedges being knocked out without disturbing the concrete, the sliding face between the two was greased. By removing the wedges, the panel was allowed to fall clear of the bottom of the launder.

There were two inside and two outside panels for the side forms. The outside panel was in the form of a channel, the vertical portion was built of 1-in. shiplap, cleated with 1 × 2½-in. pieces at 1½-ft. centers, nailed alternately with the 1-in. side *I*, and the 2½-in. side *J* against the panel; the top and bottom flanges *H* were of 1 × 10-in. shiplap fastened by triangular pieces to alternate cleats. The inside panels of the 1-in. shiplap had similar alternating stiffeners *K* and nailing strips but no flanges, since 1 × 2½-in. cross-braces *P* connected each set of stiffeners of the four panels, being bolted to them. Along the inside face of the inside panel was nailed a 1 × 1-in. strip *L*, to form the recess for the top slabs. The forms were made as light as possible so that they could be put in place by one man, although to remove the sides at the end of 24 hr., it was thought wiser to use two men.

The outside panels were set on 6 × 8-in. timbers *M*, on top of which 2 × 4-in. pieces *N* were laid inside the side panels, to form the bottom of the ridge along the bottom of the launder. By means of the wedges, the height of the bottom panel was regulated so that the tops of the planks were just an inch higher than the tops of these 2 × 4-in. pieces. After the bottom forms were properly adjusted, the outside panels were secured against spreading by toe-nailing them to the 6 × 8-in. pieces, the nails being driven only part way so that they could be pulled without disturbing the concrete.

The spreading of the inside forms was prevented by braces of ¼ × 1-in. iron *O* bolted to the stiffeners of the inside panels at their bottom ends and to the cross-braces above. Two holes were provided in the upper ends of the braces; with the bolts in the upper of these, the side panels were in their proper place; when it came time to lift off the side forms, the upper bolts were taken out, the inside panels swung in and the bolts put in the lower of the top holes, thus preventing the side panels from swinging and possibly battering the sides of the launder while being lifted off. It took about ¾ hr. to pour one section.

The completed launder was carried by the columns of reinforced concrete that supported the conveyor house. These were spaced at 16-ft. intervals, and were 16 × 16 in. in section, their corners slightly beveled, as shown in Fig. 151. They were reinforced with a ½-in. strand of wire rope in each corner, and rested on a concrete pedestal buried about 5 ft. in the ground. These columns were from about 7 to 24 ft. high above the ground.

The cross-saddles for carrying the launders were put in to grade, allowing an inch in height for lining in. These saddles were built up from a 10- or a 12-in. I-beam, according to what could be found in the scrap pile. The beams were anchored in the column by riveting to 3-in. angles set vertically in the column, the flanges on one side of the beam being sawed away to allow the web to lie flat against the angle as shown.

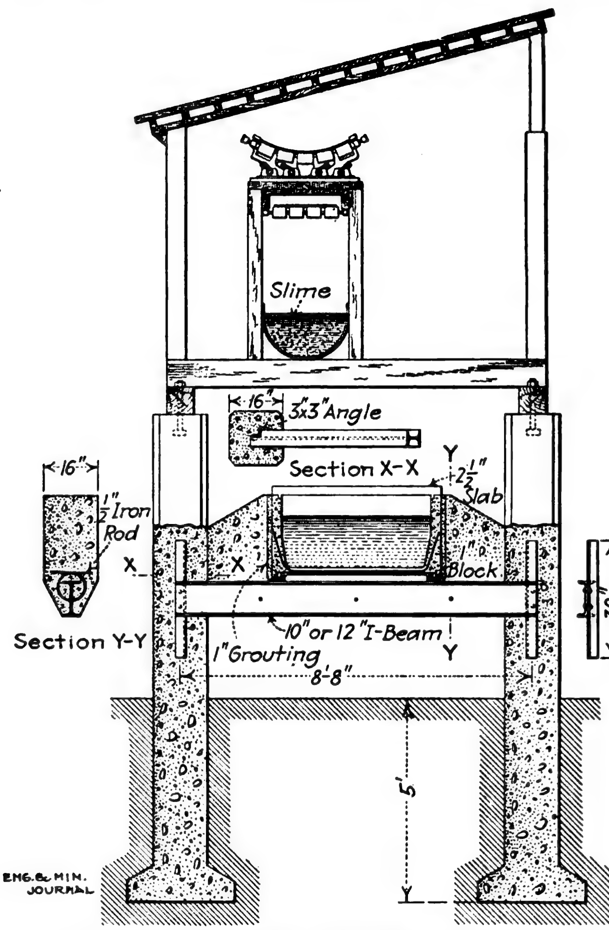


FIG. 151.—SUPPORTING THE ERECTED LAUNDER.

The conveyor house was 10 ft. wide and the distance between post centers, $8\frac{1}{2}$ ft. A good deal of concrete was therefore necessary to fill in between the posts on top of the beam so as to grip the launders. About 2 in. of side play was allowed for. This filling of concrete was made to flare from the bottom of the I-beam so as to attain a width of 16 in., and was carried high enough to come even with the top of the covering slab.

In order to anchor this concrete securely to the I-beam, three holes were bored through the beam and through each of these was threaded a $\frac{1}{2}$-in. rod bent into the shape shown in the cross-section of the saddle. The sections were butted against one another in the saddle and set to grade by means of level pieces. Then a grouting of one part of cement to one part sand was poured in around them, securely binding them to the supports. This concrete launder was devised by W. H. Schacht, assistant manager of the Champion Copper Co., while in charge of the Baltic mill.

Marking Launders for Mill Solutions (By John Tyssowski).—In concentrating mills it is always advisable to have launders marked plainly, so that the millmen and laborers can immediately tell what each line carries without following it to its source to determine this fact. In the new mill of the Bunker Hill & Sullivan company, at Kellogg, Idaho, this is accomplished by painting in various colors the launders and pipes carrying the different pulps and solutions, the same color being used on all launders throughout the mill that carry the same material. Pipes and launders for concentrates are painted red throughout the mill; those conveying middlings, yellow; slimes, white; and tailings, gray. This is especially advisable where ignorant or untrained laborers have to be employed as is usual about most mills. It is easy for almost any person to keep in mind the significance of the colors, but it might be impossible to follow through the mill and tell whether a certain pipe carried concentrates or tailings. In any case, pipes and launders should be painted, and it is little more expensive to use the different colors and this is a decided advantage, as explained.

A Pocket to Prevent Launder Wear.—A launder which conveys products from jig and classifier spigots usually wears through in a short

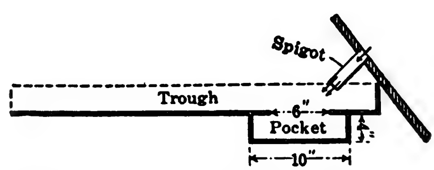


FIG. 152.—LAUNDER WITH SAND POCKET.

time at a point where the sands from the spigot strike the bottom of the launder. A scheme that is used in some of the Joplin mills for saving this wear consists of providing a pocket about 4 in. deep, 10 in. square, with an opening about 6 in. square connecting it with the bottom of the trough. Fig. 152 shows the usual construction. Concentrates accumulate in this pocket, thus affording a protection from the abrasive effect of the discharge.

Prevention of Foaming in Launders (By Victor H. Wilhelm).—A simple device to prevent the foaming-over of slime launders is used in the Last Chance cyanide plant, at Mogollon, N. M. It was usual for foam to collect in the Dorr classifiers and overflow the slime launders, especially when there was much wood mixed with the ore. This overflowing was successfully overcome, by placing a small wooden paddlewheel in the launder, as shown in Fig. 153. This wheel was revolved by the force of the slime stream, the paddles serving to knock down the foam.

Aids to Launder Efficiency.—Flow in launders is aided if the corners are filled with triangular strips of wood, and that portion wetted by the stream of pulp lined with some smooth, abrasion-resisting material such as rubber belting ("Rand Metallurgical Practice, Vol. I," p. 104). More recent plants on the Rand have put down concrete launders on the

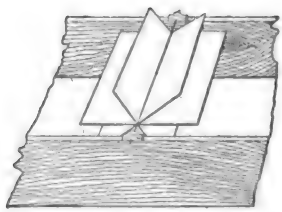


FIG. 153.—WHEEL TO PREVENT FOAMING.

solid ground where possible. When the launders are overhead, U-shaped sheet-steel, concrete-lined launders are erected. In each case the wetted surface is finished off with cement to give a smooth surface. Bends where unavoidable should be wide sweeping and given additional fall, and should not be placed too near in front of a distributing point. Distribution immediately after a change in direction of the pulp is always unsatisfactory, as unequal consistency of pulp in the subdivisions results.

Launders in Concentrating Mills.—The ordinary launder in concentrating mills is a wooden trough, lined with sheet steel. When the lining wears through, which quickly happens, the trough becomes leaky and a general nuisance until it be relined. Sometimes it is useful to substitute for the ordinary launder a 3-in. or 4-in. wrought-iron pipe. When a hole wears in this, the pipe may be turned 90 deg., or as much as necessary, and this may be continued until the pipe is so far gone that it has to be relegated to the scrap heap.

DEWATERING DEVICES

Dewaterer for Jig Tailings (By John Tyssowski).—The dewatering device shown in Fig. 154 is adapted from the type sometimes used to

handle the product from log washers. It was formerly used, in the mill of the Bertha Mineral Co., at Austinville, Va., to handle the tails from a Hancock jig and gave good results when operated with fairly coarse material; a large proportion of slimes or mud caused too great a consumption of power. The details of construction are shown in the drawing. The dewaterer is fed from the side, the material being scooped up as the frame turns and delivered over the discharge lip. The buckets being made of perforated metal allows the material to drain as it is ele-

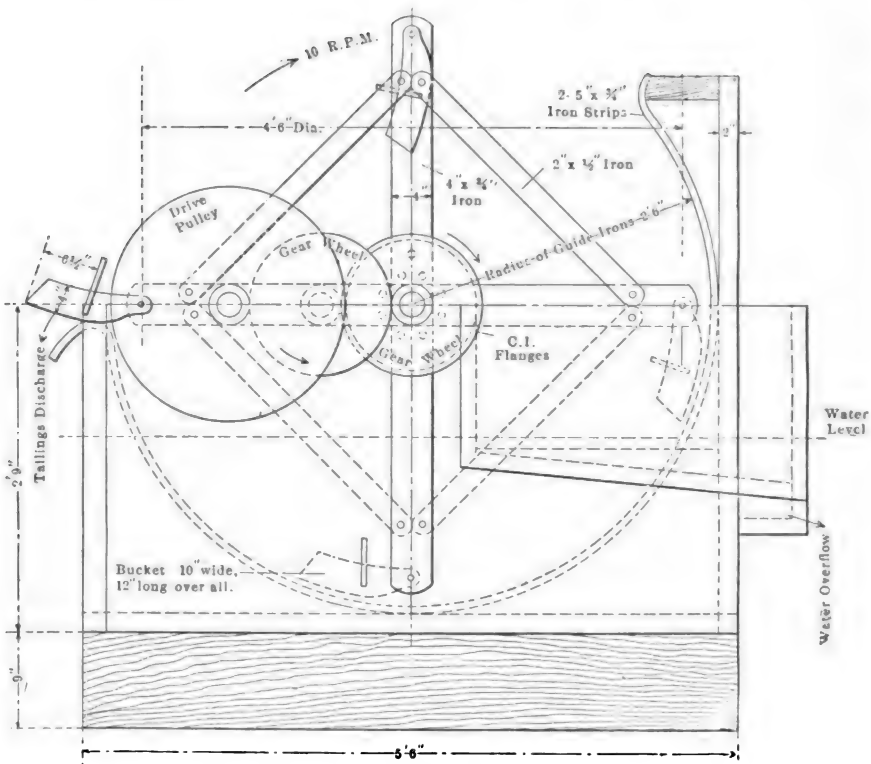


FIG. 154.—DEWATERER FOR JIG TAILINGS.

vated. Run at 10 r.p.m. the apparatus has a capacity of about 4 tons per hour.

Tailings Dewatering Wheel.—In the drawings given in Fig. 155 are shown the details of construction of the dewatering wheels 14 ft. in diameter that are used in the No. 3 mill of the Doe Run Lead Co. at Rivermine, Mo. These wheels handle the tailings before discharging them on the belt conveyor that takes them to the tailings pile. There are two of these wheels; each handles 1500 tons of pulp in 24 hr. when run at 2 r.p.m., the most efficient speed.

Referring to the general drawing of the shovel or dewatering wheel, it will be seen that this wheel is made of segments tied together by bolts and arms to a cast-iron hub. This hollow hub-casting extends beyond the sides of the tank that the wheel runs in and forms the axle to which the driving gear is keyed. The hub has sockets in it to receive the pieces of

FIG. 155.—SMALL TAILING DEWATERING WHEEL USED IN FLAT RIVER DISTRICT, MISSOURI.

4-in. pipe that are used for the spokes or arms of the wheel. There are eight of these spokes or arms, and they screw at their outer end into a cap piece that bolts to flanges on segments of the rim. There are eight segments in the rim which are bound to the hub by $\frac{5}{8}$-in. tension rods *R* that

go through lugs on the rim segments and hook into the hub. These tension rods are hooked into the hub before the rim is assembled.

As will be noticed the tension rods alternately pass through the wheel from different sides, and each rod hooks into the hub on the other side from the lug on the rim into which that rod fastens. The arms at the inner end are fastened to the hub by keys. After the wheel has been assembled and trued, zinc is poured around the pipes where they fit into the sockets of the hub. To these outer segments, sheets of $\frac{1}{4}$-in. iron 18 in. wide are fastened to form a deflector for catching the rock particles that adhere to the shovel blades.

The shovel blades or scrapers are then bolted through the rim of the heavy reinforcing segments of the wheel. These scrapers slant back slightly from the direction in which the wheel rotates, so as to make the angle of the discharge about 45 deg. They are cast with $\frac{5}{8}$-in. holes so as to allow the water to drain back out of the tailings as they are scraped up the trough. This, to protect it from wear, is fitted with cast-iron wear plates. The scraper blades are 18 in. wide and are made concave so as to cause the tailings to run toward the center of the shovel, as the shovels rise out of the water narrowing the stream that is poured out into the spout that leads to the conveyor belt.

The tailing wheel rotates in a concrete tank with sloping sides so that any tailings that settle on the sides will slide down to the bottom of the tank and be scraped up to the discharge by the shovel blades. In the tank is a trough carried on iron bars that are anchored in the concrete. The purpose of the trough is to confine the coarse tailings that settle into the shovel box so that the blades can scrape them up to the discharge spout easily and without spilling. The trough does not extend much beyond the point where the tailings are poured into the settling tank by the feed launder that comes in from the side of the box. The feeder launder comes in at about the level of the discharge spout while the overflow lip is about 8 in. below the bottom of the discharge spout.

In the first wheel built, the false bottom or trough was carried directly under the scraper blade so that there was not much clearance left for the blades. Some of the blades became broken in one way or another and these fragments upon getting wedged between other blades and the false bottom broke more blades. Consequently in the later wheels, the trough is being placed lower so as to allow more clearance around the shovel blades. But it is probable that it would be still better to let the tailings built up their own bottom in these shovel tanks.

These tailing wheels are driven by gears, and the driving pulleys of the train are arranged so that the speed of the wheels can be varied so as to run at 4, 2 or 1 r.p.m. This is in order to determine the best

speed at which to run the wheel so as to get the lowest amount of moisture in the tailings being handled.

Dewaterer for Jig Tailings (By John Tyssowski).—The screw conveyor shown in Fig. 156 has given satisfactory results as a dewaterer for

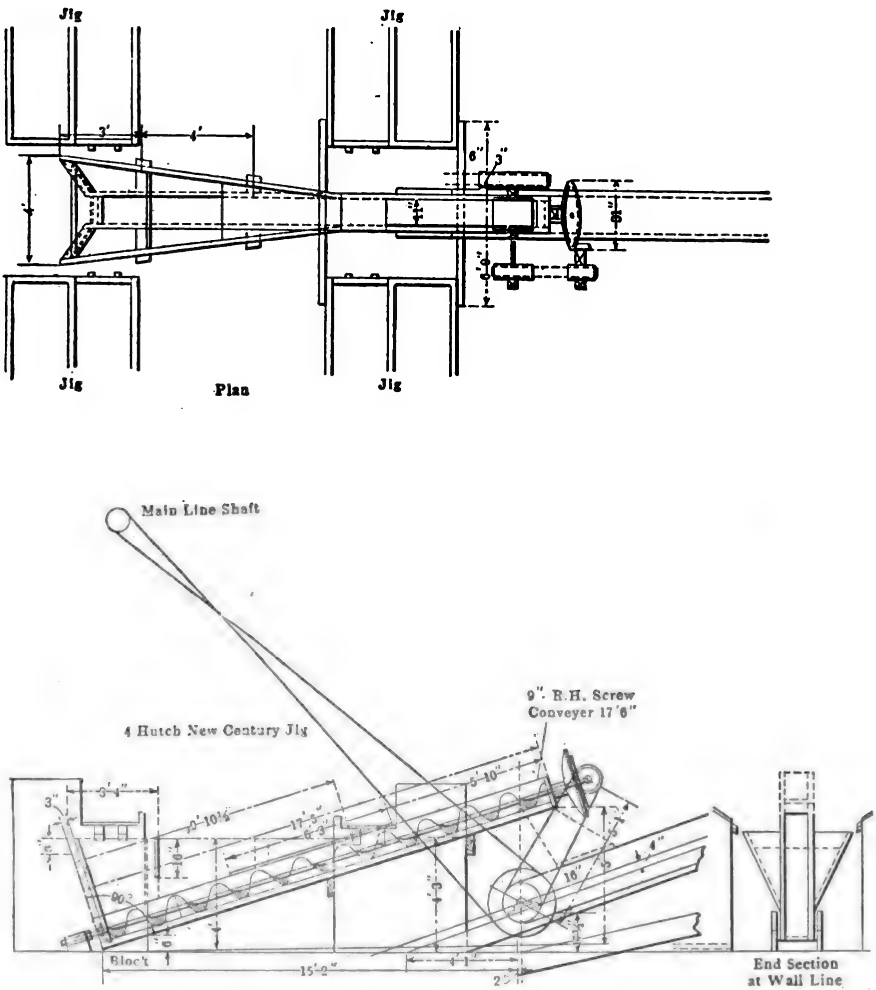


FIG. 156.—METHOD OF SETTING UP SCREW CONVEYOR AS DEWATERER.

jig tailings at the hard-rock mill of the Bertha Mineral Co. at Austinville, Va. At this mill various screening and other dewatering devices were tried and either proved unsatisfactory or had to be abandoned on account of excessive wear. The screw conveyor, however, has been found to deliver jig tails containing practically no free water, and the water

from these tails contains only slimes. Satisfactory results were obtained on material as fine as 30 mesh operating the conveyor at speeds varying from 10 to 50 r.p.m. The conveyor handles the tailings from four Harz jigs, and with coarse material the power consumption is quite small. In the same mill screw-conveyor dewaterers set vertically have been tried to handle various products. Using a smooth-bore cylindrical shell about the screw, difficulty was experienced from fine material slipping through and clogging the space between the screw blade and its inclosing tube. This difficulty has been overcome by opposing the screw blade by a

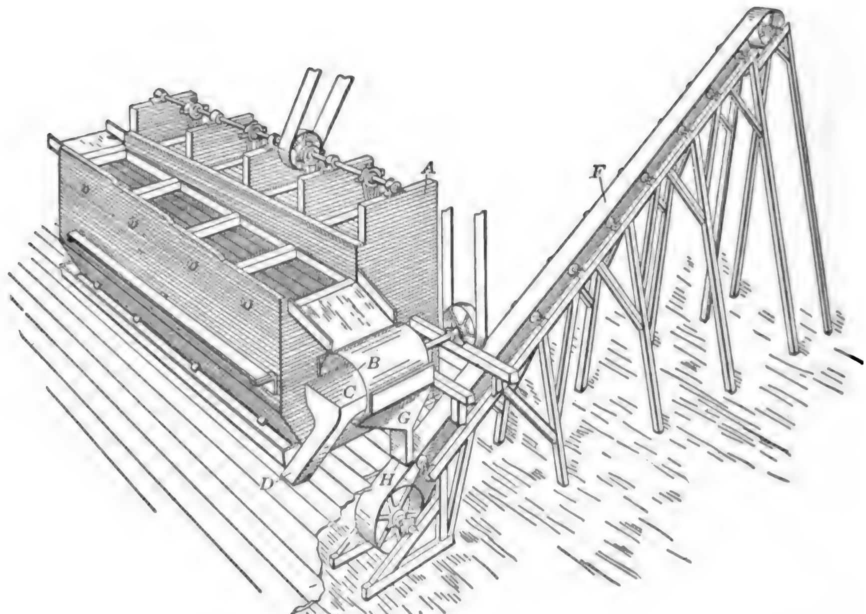


FIG. 157.—DEWATERING DEVICE AND BELT CONVEYOR FOR JIG TAILINGS.

narrow metal strip riveted to the inside of the tube so as to form a sort of rifling. Using a tube 8 ft. high and 9 in. in diameter and rifled in this manner, material of jigging size can be successfully elevated 4 ft. above the water level without excessive power consumption. An ample opportunity for water to drain off is thus afforded and the material delivered is so dry and compact that no water can be squeezed from it by the hand.

Method of Handling Slimes and Tailings (By A. O. Ihlseng).—The accompanying sketch, Fig. 157, gives a view of the general arrangement of a device for dewatering tailings and disposing of them in a more economical manner than has heretofore been the practice in the Joplin district. I have always considered that the method of handling tailings from the mills could be improved, and that the losses of zinc and lead in

the fines and slimes carried away were too high, and that some method of separating the water and fines from the tailings should be utilized. This also left the matter of tailings disposal to be solved.

While manager of the Oronogo-Circle properties I found an excessive loss in fines. After considering the various screens and devices in use for separating sludge, as the fines and slimes are called in this district, I decided to experiment on simpler lines. I installed at the end of the jig a revolving screen, *b*, 36 in. wide and 30 in. in diameter, of 1 mm. mesh. This screen is supported at one end by a shaft, and is driven 10 r.p.m. by a belt from either the jig shaft or any convenient shaft. The pan *c*, suspended in the screen, is supported on the outside. The water and tailings drop on the screen, *b*, and the tailings fall into the hopper *g*, comparatively dry. The water and fines drop in the pan *c* and are carried by the trough *d* to settling tanks, where the slime is allowed to settle for future treatment. The screen is exceedingly simple, efficient and easily kept clean. The separation is perfect.

The disposition of the dry tailings was then the problem to be solved. At first the tailings were supplied with water from the slime tables to be carried up by the ordinary tailing elevator and distributed to the waste pile. This method of distributing tailings is crude and unsatisfactory in many ways, though its simplicity and crudeness are its chief recommendations. Having the tailings dry, a means was looked for to dispose of them readily. The use of a car and incline, such as is used in handling slate and waste at the coal mines, was tried. This was found to be satisfactory only to a limited extent, as it required labor and attendance.

Shortly after severing my connection with the Oronogo-Circle company I erected one of the largest mills in the district at Galena, Kan., for the Eureka Mining Co., in which I was personally interested. I was determined to provide more economical handling of the tailings, and adopted a 14-in. belt conveyor, something which had never been suggested or attempted in the district. The conveyor has a capacity of 60 tons per hour when traveling 150 ft. per min. The tailings from the screen fall into the chute *h*, and on the conveyor belt *f*. The belt runs on concave rollers placed on the frame. The tailings are dropped at the head of the belt until the pile accumulates to a height where other distribution is required. A 2-in. centrifugal sand pump takes the tailing water from the slime tables, and pumps through a 2-in. pipe laid on the side of the conveyor frame. This stream of water enters the box into which the tailings from the belt fall, and the mass flows out in the ordinary iron-trough tailing distributors. The arrangement shown in the illustration accomplishes two distinct purposes: It saves the fines, and handles the tailings cheaply.

The experience in this district in handling tailings makes this form

of distribution attractive. The usual tailing elevator of a 300-ton plant costs $1000 for frame, 20-in. belt, elevator cups, troughs, etc. Such an elevator is 60 ft. high, and delivers the tailings 80 ft. from the mill at an actual height of 30 ft. It requires about 5 hp. to operate. In winter the water freezes, and the outside troughs break down with the weight of ice, and frequent accidents occur to the men working on the troughs and head of the elevator. The belts and cups of the elevator must be renewed every six months, or less. Each renewal entails a cost of $350, exclusive of labor. The elevator belts break down frequently as they become worn. The actual loss by stoppage and breakdowns of these elevators amounts to $2500 annually, making the actual maintenance expense run up to $3000 or $3500 annually as a minimum.

Compare the above with the belt conveyor. The entire conveyor costs $500 to install. It will run two years with but little repair. There is nothing to get out of order. A renewal belt costs $125, and can be put on in two hours. A belt conveyor will deliver the tailings at a height of 32 ft. at the same distance from the mill the tailings elevator delivers its tailings. This is a gain of 2 ft. Further, the low structure eliminates all accidents. The conveyor can be extended indefinitely, while the only method of extension in the ordinary tailings elevator of the district is to build an auxiliary, or "dummy," on top of the tailings pile.

Dewatering Tailings.—A system of dewatering tailings was recently installed at the Oronogo Circle mines, Joplin district, Missouri, which is essentially as follows: The coarse chats from the roughing jigs are elevated and passed through a 36×48-in. trommel. The screen used is 1½ mm. The water and fine sand pass through the screen and return to the slimes, while the coarse material, minus the water, goes into a storage bin. The sands and slimes from the sand jigs pass by another elevator to a second trommel, 36×120-in. placed above the same receiving bin. The screen on this trommel is ¾ mm. The sand size from this trommel is delivered into the same bin with the coarse chats, from which it is emptied into cars and deposited on the waste dump. The slimes and fine sand that pass through the second trommel go back to classifiers to be distributed upon tables for further treatment. By dewatering the tailings in this way they may be stacked much higher without encroaching upon adjoining property. The slimes from the tables are run into large settling ponds.

Stationary Dewatering Screen (By Claude T. Rice).—Four stationary screens inclined at an angle slightly greater than the angle of repose of the material are used in the Leadwood mill of the St. Joseph Lead Co., in the Flat River district of Missouri, for dewatering the tailing from Hancock jigs before it is discharged upon a conveyor belt. Each screen is 24 in. wide by 28½ in. long and is held in place in a stationary

frame by 1-in. cleats on each side. Across the lower end and on the upper surface of the screen a cleat is nailed that holds a thin layer of tailing on the screen's surface, which layer protects the wire cloth from excessive wear. The capacity of each screen is about 875 tons per 24 hr. Tyler No. 58 woven-wire, or No. 18 cold-rolled, hand-punched, staggered long-slot, steel screens are used. The slots of the latter are 0.058 in. wide by 11/16 in. long. The wire screens cost $1.94 each, last 14 days and screen 12,250 tons of tailing, while the punched screens cost $1.65 each, last 20 days and screen 17,500 tons. The screen may be set at an angle as low as 26 deg. and the tailing will move over it. The observation has been made that these screens, if movably supported and given a slight jerking motion in a forward direction by an eccentric, could be set at a much lower angle, and the capacity and possibly the life be increased at a nominal expense for the power required. Shaking screens of this type have been used successfully in other mining districts, but only for sizing dry ore. When used for dewatering thin pulp the motion should prevent blinding.

Handling Concentrates at the Daly-Judge Mill (By Claude T. Rice).—The Daly-Judge mill at Park City, Utah, makes three classes of concentrates, a lead, a zinc and an iron-zinc product. These concentrates are handled automatically in a clever manner. The con-

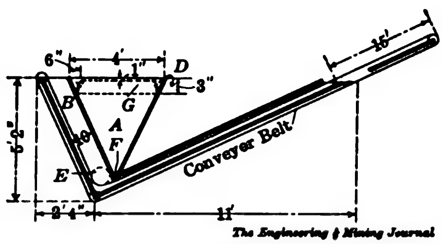


FIG. 158.—CONCENTRATE DEWATERER.

centrates coming from the different tables flow to their respective launders, which are made of wood and lined on the bottom with galvanized iron. A grade of only 8 in. in 150 ft. is given to the launders, but as they are given a violent shake by means of the Wilfley head motions, having a 1-in. stroke, that are attached to their head ends, the concentrates travel rapidly along them. The three launders discharge to their respective bucket elevators.

Referring to Fig. 158, each elevator discharges into a settling box A, having a top baffle G. This box discharges the settled concentrates continuously to an inclined traveling belt that operates in an auxiliary box B that extends at each end to a height even with the top of the settler so that the water in the concentrates, while it works down with them into

the conveyor box, can discharge only over the lip of the settler at *D*. One side of the conveyor box, *B*, hugs the side of the settler while the other puts off at an angle of about 24 deg. from the horizontal. Two of the rollers around which the conveyor belt runs are above water, while one, *E*, is placed so as to cause the belt to pass close under the discharge opening *F* of the settler box.

The long end of the belt extends about 15 ft. beyond the level of the water in the conveyor box so that the concentrates have a good chance to drain as the belt is run at a speed of only about 50 ft. per min. At the upper end the concentrates discharge into a bin with about 8 per cent. moisture. Experience has shown that the belts of these dewatering devices should have a lower slope and that the part out of water should be longer so as to get the best results. The overflow from the settler goes to a Callow cone and after being settled again, so as to guard against the loss of any slime concentrates, is returned for use on the tables. Four-ply rubber belts 20 in. wide are used in these dewatering devices. One of them will handle 17,000 tons of concentrates before it wears out.

Friction Loss in Wrought-iron Pipe.—Due to widely varying tables that appear in different handbooks and the necessity of elaborate interpolation to make them usable, a great deal of time is generally wasted in solving problems in which friction of water in pipes is a factor. Ira N. Evans (*Power*, July 9, 1912), has taken six of the best formulas for the friction of water in commercial wrought-iron pipe, together with the results of several tests, and compared them in tabular form.

The following were the principal formulas used by Mr. Evans in obtaining his average:

(1) Williams & Hazen formula:

$$V = cr^{0.63}\left(\frac{h}{l}\right)^{0.54} 0.001^{-0.04}$$

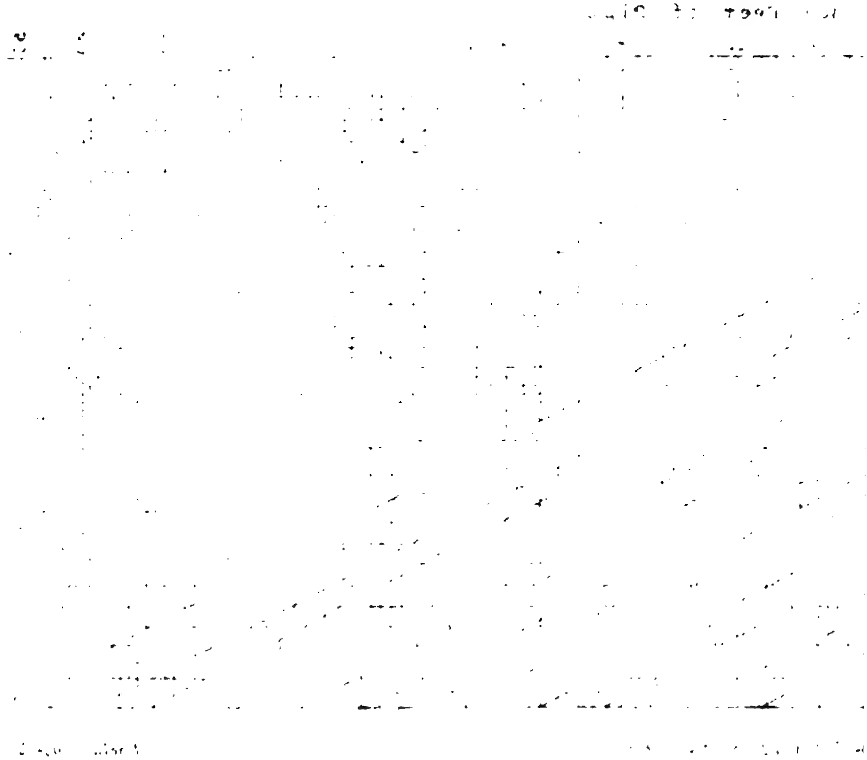
V = Velocity in feet per second;
c = Constant varying with condition of the pipe;
r = Hydraulic radius $= \frac{d}{4}$ in feet;
h = Friction head;
l = Length of pipe in feet.

The constant 120 gave results for most of the sizes higher than the average, while the factor 125 would give results checking closely with the average.

(2) Darcy's formula:

$$h = 0.003732\, lV^2 \frac{D+1}{D^2}$$

D = Diameter in inches.



This formula gives values higher than the average.

(3) Meier's formula:

$$h = 0.0038\, l \frac{V^{1.86}}{D^{1.25}}$$

D = Diameter in feet.

(4) Harrison's formula:

$$h = 0.00375\, l \frac{V^2}{D}$$

D = Diameter in inches.

The results from this formula are below the average for pipe up to 6 in. in diameter.

(5) Fanning's formula:

$$h = f \frac{lV^2}{d2g}$$

The constant f varies for each change in size and velocity. This formula follows the average closely and gives results more consistent than the others except that of Williams & Hazen.

(6) Serginsky's formula, compiled from the latest German authorities, was also used. If gives results above the average for small sizes and below the average for large sizes.

The average values derived by Mr. Evans, mainly by the foregoing formulas, have been plotted by W. L. Durand, on logarithmic paper, as shown in the accompanying curves (*Power*, Oct. 29, 1912). All equations of the general form $y = Bx^n$ will plot at straight lines. Several of the formulas are of this form, and the others vary from a straight line to only a slight extent. The average values then may safely be represented by a straight line, as the greatest variation will be much less than the difference between any two values that make up the average.

The method of using these curves is self-evident, there being three variables and in any problem containing two of them the third can be found immediately from the chart. If only one variable is known, several values of one of the other two may be assumed and the corresponding values of the third found from the chart given in Fig. 159. The values that seem to best fit the problem may then be chosen.

The chart is plotted with velocities from 1 to 10 ft. per sec. as abscissas and friction head per 100 ft. from 0.1 ft. to 100 ft. as ordinates. Each curve is for a definite pipe size as indicated, and with a little variation, all the way through for velocities of 1 ft. per sec, the chart shows values practically identical with the averages in Mr. Evans' table.

For example, take a 3-in. pipe with a velocity of flow of 6 ft. per sec., the friction head per 100 ft. of pipe would be 5.45 ft. To produce a drop in head of 2 ft. in 100 ft. of 5-in. pipe would require a velocity of 4.7 ft. per second.

A Convenient Rule for Pipe Sizes.—L. B. Lent, according to *Engineering News*, gives the following: The velocity of flow in water pipes is very commonly taken at 3 ft. per sec. This does not produce excessive friction loss even in comparatively small pipes. With this velocity: Diameter of pipe (in inches) is equal to the square root of the quantity of water flowing (in cubic feet per min.). The rule is not exact, but involves an approximation of something less than 1%. The rule, of course, merely expresses the relation between area, velocity and volume of flow, and involves no hydraulic relations or laws. The rule can be used with gallons per minute after dividing by 7.5 (more exactly 7.48) to get cubic feet per min. It can be used with a velocity of flow other than 3 ft. per sec. by first solving the problem for the 3-ft. velocity, then dividing by the square root of the correct velocity and multiplying by the square root of 3.

Hanger for Light Pipe.—The general details of a convenient pipe hanger for supporting light pipe, from 1 in. up to 3 in. in diameter are shown in Fig. 160. It is made up of a V-shaped $\frac{3}{4}$-in. band-iron $\frac{1}{16}$ in.

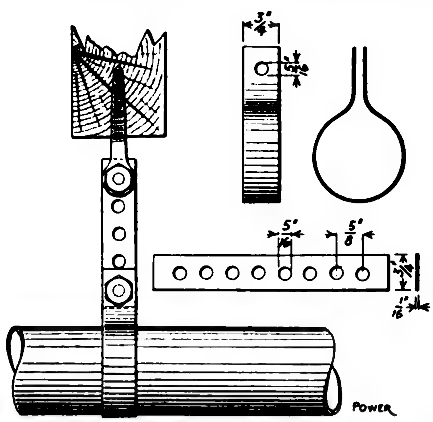


FIG. 160.—HANGER FOR LIGHT-WEIGHT PIPE LINES.

thick, and bent so that the inside diameter is the same as the outside diameter of the pipe it is to support. The flat ends of this piece are punched with $\frac{5}{16}$-in. holes, says H. M. Nichols (*Power*, Oct. 22, 1912). The second part of the hanger is a flat strip of band-iron in which a number of $\frac{5}{16}$-in. holes have been punched about $\frac{5}{8}$ in. apart. The strips can be made up in random lengths and when used broken off to the desired length. Eye lagscrews are used to fasten the pipe hangers to the ceiling. These lagscrews are most easily put up before the pipe is secured in place. The bands can be put on the pipe and clamped loosely to the straight strips with $\frac{1}{4}$-in. bolts. After the pipe is lifted into place it is only necessary to bolt the straight strips to the lagscrews with $\frac{1}{4}$-in. bolts. This type of

pipe hanger is particularly convenient when it is necessary to run a line of pipe on a slant.

Simple Pipe Clamp.—A simple and effective clamp for water, steam or air pipe is described by E. Oster, in *Power*, Feb. 11, 1913: The device is of wrought iron and for pipe under 2½ in. should be slightly less than 1/16 in. thick at the bottom, so as to be somewhat flexible, gradually increasing, as shown in Fig. 161, to ⅛ in. in thickness at the base of the projections or lugs, at the top. The latter should be about ¼ in. thick, have a height and width of 1 in., with a hole in the center to take a ⅜-in. bolt. The greatest width of the clamp, at the extreme bottom, should be about 1¾ times the nominal diameter of the pipe for which it is made. The clamp should be

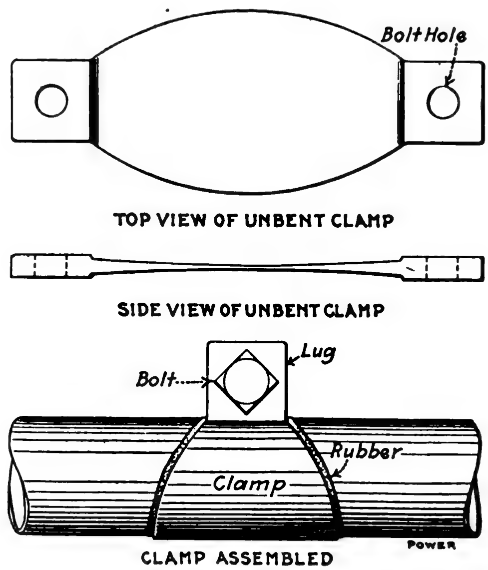


FIG. 161.—HOMEMADE PIPE CLAMP.

made so it will be in contact with the pipe through an arc of about 300 deg. A piece of cloth insertion rubber, at least 1/16 in. thick and a little larger than the clamp, should be placed between the leak in the pipe, and the clamp. The clamp can easily be made by a mine blacksmith and is capable of application to leaks otherwise almost inaccessible and without reduction of the pressure on the system.

Gate Valves for Battery Pipes (By C. W. Walker).—Experience has taught me that a gate valve is always preferable to a globe valve in mill work. Globe valves give trouble from dirt, trash and moss accumulating under the seat, which makes removal of valve necessary for cleaning, while with a gate valve the dirt passes through when fully opened. It is better to use as small a valve as will give the required amount of water

under the lowest pressure, for a small valve almost fully opened is less apt to be clogged than a larger one only partly opened.

DISTRIBUTORS

Barrel Distributor for Concentrating Tables (By John Tyssowski).—The scheme used in the new Bunker Hill & Sullivan mill, at Kellogg, Idaho, for distributing pulp to the Frue vanners is extremely simple and satisfactory. The distributors are merely barrels suspended about 5 ft. above the table tops, from which pipes radiate to the feed troughs of the concentrating tables. The distributors are suspended above the aisle between rows of vanners by iron rods fastened to the ceiling. The barrels are 1½ ft. in diameter and 2 ft. high. Each is tapped at a point at the center of the bottom, for a short length of 2½-in. pipe from the feed pipe carrying pulp, which is run below all the distributors. Inside the barrel there is a 6-in. central pipe (extending nearly to the top of the barrel) over the top of which the intake solution flows into the outer compartment from which the outlet pipes discharge. The outlet pipes are 1½ in. in diameter and radiate from the barrel as shown in Fig. 162. One barrel

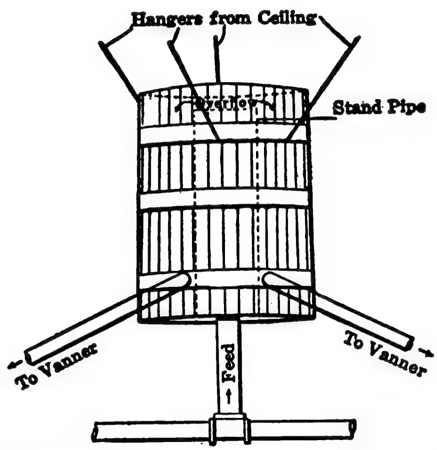


FIG. 162.—FEED DISTRIBUTOR FOR CONCENTRATING TABLES.

serves to distribute pulp to four or six tables, and the desired regulation of feed can be had by placing the barrels at the proper elevation above the concentrating tables. This is one of the simplest distributors for feeding concentrating tables that is in use in the mills in the country, and it is claimed to be quite satisfactory in its operation. Distributors of similar design are also used with satisfactory results at Stratton's Independence mill in the Cripple Creek district.

The Kidney Pulp Distributor (By Claude T. Rice).—The one completed section of the Ohio Copper Co.'s mill at Lark, Utah, is treat-

ing without any crowding about 50% more ore than it was designed to handle. One of the explanations for this, no doubt, is the system of making tailings on most of the tables and then cleaning the dirty concentrates from these tables on other tables where closer watch can be kept. There is thus a heavy feed of sulphide to the secondary tables so that in the riffles the smaller grains are covered by the larger ones, and so are somewhat protected from the force of the current required to wash off the larger particles of the gangue minerals. But another explanation unquestionably is found in the exceptionally even distribution of the pulp to the different tables that is obtained in this mill. In the treatment of large tonnages, especially where there is a wild attempt to load every device to the limit, as is the present tendency at many mills, it is important to have the pulp distributed equally to the different.

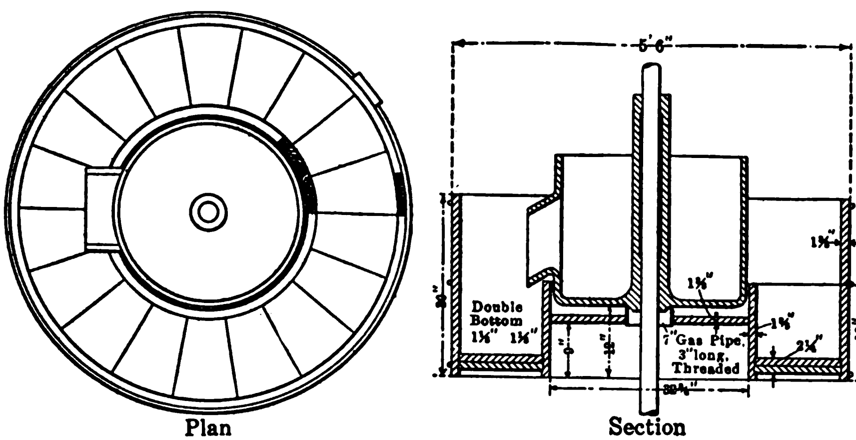


FIG. 163.—ARRANGEMENT OF KIDNEY PULP DISTRIBUTOR.

The Kidney distributor, which is used at the Ohio mill, was devised by William Kidney, superintendent and also the designer of the mill and the system of concentration used. This distributor has one drawback, *i.e.*, whenever the feed has to be shut off from one table or unit that is being fed by the distributor, the equilibrium of the feed to the other tables is disturbed, as will become evident when the construction of the distributor is understood.

The details of the design of the pulp distributor are shown in Fig. 163. The distributor consists of a rotating cylinder with feeding lip, into which is led all the pulp going to the tables or other devices that the distributor is serving. This feed cylinder stands in the center of an annular wooden tank which is divided by partitions into as many compartments as there are devices to be fed. The size of this annular tank is such that the different compartments just empty themselves during one revolution of

the feed spout. This assures an even feed to the different compartments, but in all cases at the Ohio mill a thickener or else a feed box is used before each of the tables or the chilean mills.

The feed cylinder rotates at 15 r.p.m., and necessarily, as the speed spout passes at uniform speed over the annular tank and as the compartments are all of equal size and are similar in shape, each device gets an equal and similar feed. The feed from each compartment is taken by a short 3-in. gas pipe to one of the component launders of the trunk launder serving one group of devices. By the use of these trunk launders the confusion overhead, caused mainly by the individual-launder system, is avoided.

The distributor is driven by a system of bevel gears so as to cut down the speed, as the driving belt comes from the main shafting. In order to take the wear, the revolving feed cylinder is provided with a sheet-steel lining. The cylinder itself is made of cast iron, while the annular tank is made of wood. The end of the shaft that carries the feed cylinder extends below the bottom of the tank and rests in a step bearing. The outer staves of the annular tank are 30 in. long so as to take the slop of the heavy feed which in several instances goes to the distributor. The inside staves are 16 in. high. The dividing partitions are only fastened by means of nails so that they can be changed in case it becomes desirable. All bearings are equipped with grease-cup oilers.

If for any reason it is necessary to cut off the feed to any one of the devices served, a plug is put in the discharge pipe from that compartment of the distributor. But obviously this causes that compartment to fill with pulp, and then the portion of the pulp that should go to the device which is shut down, is thrown into the two neighboring compartments, producing an overload in these. Otherwise an absolutely even feed is obtained by means of this distributor and during the nine months that the Ohio copper mill has been in operation these machines have all worked satisfactorily. They are used throughout the mill wherever it is necessary to divide the feed going to different devices. The distributors can, if necessary, be mounted one above another on the same shaft, but the necessity for doing that would seldom arise.

Since the obtaining of a maximum tonnage and efficiency of concentration from a given equipment depends mainly upon giving an equal load to each of the individuals in the unit, the use of this or some similar distributor is of great advantage in concentrating mills. Good work is impossible when one table is underloaded at the expense of an overload on another table. Cyanide men have realized this for some time and mill men are gradually appreciating this fact. The Kidney distributor is not patented and can be used by anyone without the payment of royalty.

The distributors are rather expensive to make, but they soon pay for themselves where large tonnages are being treated.

The Steptoe Distributor.—At the Steptoe concentrator at Ely, Nev., a distributor somewhat similar to those installed at Ohio Copper mill but much simpler is used. The distributor consists of a cylindrical box 30 in. diameter and 30 in. high, on the top of which are mounted the shafting and bevel gearing that drive four paddle arms, fastened to a vertical shaft, that revolves at a speed of about 15 r.p.m. On the side that the feed launder enters, a baffle board is used to keep the feed from striking the upper part of the vertical shaft and from being splashed about the mill. In the sides of the cylindrical box are fitted bushings about 3 in. diameter through which the pulp is discharged, while about 8 in. or so higher and over each discharge is another bushing to take care of any overfeeding of the distributor. Each bushing feeds its own launder which leads to a trunk launder from which the individual launders for each table branch. The revolving paddles stir the pulp into a homogeneous mixture so that the pulp issuing from each discharge opening is of the same consistency. The openings are cut at the same height, and as they are all of the same size practically an equal quantity of pulp issues from each orifice. The size of these openings is chosen so that the level of the pulp remains a few inches above the tops of the openings, discharge taking place through the full section. Of course this distributor is not so accurate in its work as the Kidney machine, but it does well enough for all practical purposes and is much cheaper to make. Moreover, if the feed has to be cut off from any table, the plugging of the corresponding orifice does not unbalance the feed of the other tables, for the amount going to each is increased proportionately.

Pulp Distribution at Homestake.—The methods of dividing and distributing pulp at the Homestake mills are described by A. J. Clark and W. J. Sharwood in *Bull.* 98 of the Institution of Mining and Metallurgy. Wherever a large stream of pulp has to be divided into equal parts for distribution to a number of parallel units, the principle adopted is to make the divisions at a single point in the wide launder and carry the fractions together in parallel streams, rather than to split off a number of successive laterals at various points from the main stream. In many cases the finer adjustment is made by tongues of cast iron. These have sharp sloping edges to meet the currents, the other ends being pivoted and the tops provided with means for clamping them in any position; the parallel divisions are of wood. Where streams unite or branch, o rmake sudden turns, a sump of wood or concrete (or for extremely small streams of cast iron) is placed at the angle, sufficiently deep to accumulate a thick bed at the bottom, which prevents wear of the material. At points where it is occasionally necessary to divert or bypass a stream from one

launder to a lower one, a "tipple" is used, made of sheet or cast iron. With the tipple in its normal flat position, the stream passes over it nearly horizontally, but when the tipple is thrown up the stream drops abruptly to a lower level. The amalgamation tailings are carried in launders variously constructed, the most satisfactory being of wood, lined with hard paving brick set closely, but without mortar or cement; the grade is 1.5 per cent. After passing the first set of cones, tailings are conveyed for

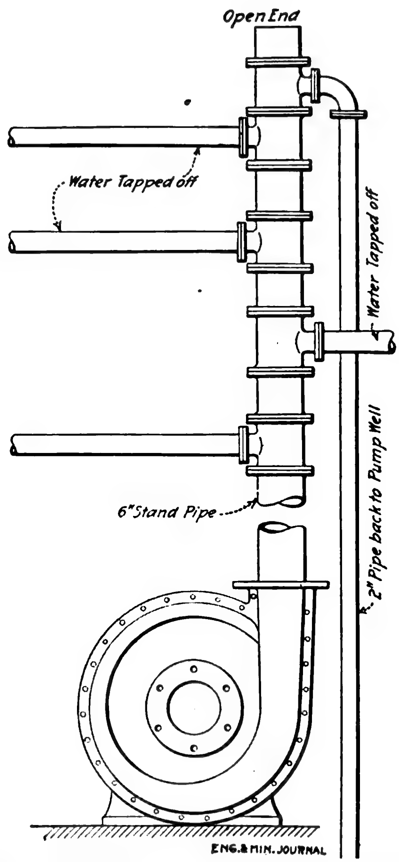


FIG. 164.—CENTRIFUGAL MILL PUMP WITH STANDPIPE.

the most part in cast-iron pipes. The present construction is to cast the pipe in a simple cylindrical form, 1 in. thick and 12 ft. long, making a butt joint covered by a loose-fitting sleeve, the space being packed with jute. With sandy tailing there is appreciable wear on the bottom, the pipe running partially full; it is customary to turn the pipe through an arc of 120 deg. when wear has progressed to a certain extent, and a pipe will thus be turned twice before being rejected as worn out. The turn-

ing of a pipe may be accomplished rapidly and without shutting off the flow of pulp.

PUMPS

Standpipe on a Mill Pump.—In some of the mills at Joplin, Mo. the pressure on the hydraulic classifiers and on the water going to the jigs and tables is regulated by small supply tanks into which the mill pumps raise the water and from which it goes to the different devices throughout the mill. This system has its drawbacks because when suspending operations, the water has to be shut off by cocks which are usually in some unhandy place or else allowed to run until the supply tanks are empty with the result that the wells of the elevators become flooded. Moreover in case the valves are closed time is lost in making adjustments to re-establish the desired flow. In winter during freezing weather trouble is experienced where tanks are used because the mill is operated but two 10-hr. shifts per day and while the mill is not in operation the water in the pipes may freeze solid. To avoid the inconveniences experienced with the supply-tank system, J. G. Marcum, a mill designer of Joplin, Mo., uses a standpipe immediately above the pump and takes water from it at the heights corresponding to the pressures that he desires. This arrangement is shown in Fig. 164. The standpipe is about 6 in. diameter, but the size depends upon the quantity of the water required in the mill. The top end of the standpipe is open and is provided with a small pipe to convey any excess water overflowing at the top back to the pump-well or the suction end of the pump. The speed of the pump is regulated so that there is only a slight overflow yet enough to insure a constant pressure in the mains. When it is desired to suspend milling it is only necessary to admit air to the pump and the pipes and valves are all drained of water so that they cannot freeze. In the morning it is only necessary to start the pump after shutting off the air.

Automatic Pump Control.—In mining practice it is not always convenient to discharge the water used in cooling the jackets of air compressors, where it will run to some place out of the engine room. In one instance it was necessary to discharge this cooling water into a tank under the floor of the compressor room. In order to keep the water level in the tank uniform, a pump was used, which discharged the water at a distant point. The steam pipe of the pump was fitted with a globe valve, the stem of which carried a sprocket wheel. A sprocket chain ran over this wheel, one end being attached to a suitable weight and the other to a float in the tank which was directly under the pump. As shown in Fig. 165, as the height of the water increased or decreased, the float opened and shut the valve, so that the water level was kept practically constant. In Fig. 166 is shown an arrangement devised by E.

T. Lehman (*Power*, Mar. 5, 1912), to automatically maintain a constant level in a storage tank on a roof. The small pail *A* controls the steam valve of the pump. The overflow from the tank empties into the pail, causing the valve to close. A small hole in the bottom of the pail allows the water to drain out. When no overflow is falling into the pail the water drains out and the weight on the valve lever overbalances the weight of the pail, opens the valve and starts the pump. The makeup water is automatically supplied by the pipe *B*, which is controlled by a float valve, as shown.

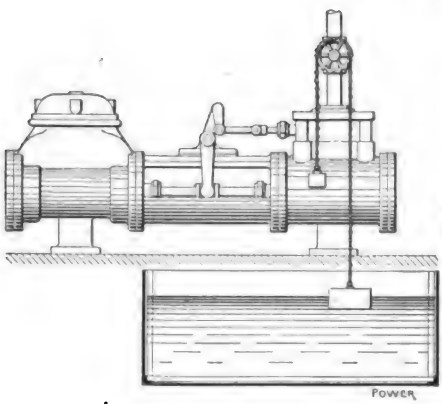


FIG. 165.—VALVE OPERATED BY A FLOAT.

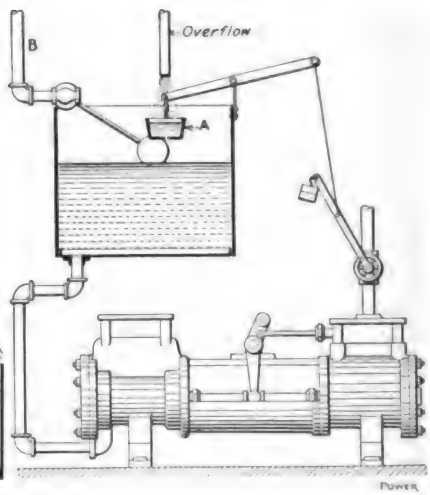


FIG. 166.—AUTOMATIC PUMP CONTROL.

Pump Suctions from Cyanide Tanks (By H. T. Durant).—Centrifugal pumps when used to pump from tanks are usually, where possible, arranged to work fully "flooded," thereby entirely avoiding suction lift, the necessity for priming, and the use of foot valves, all of which are a source of delay when endeavoring to run full time. Sometimes the flanged suction connection is made to the bottom of the tank, and at other times to the side, as near the bottom as possible. Unless some device is used by which the pump can practically empty the tank before it takes enough air to prevent further work, it will be found that, in the case where the suction connection is on the side of the tank, several inches of water still remain in the tank after the pump ceases to draw. Those who have had to clean out a large tank which has no other exit for mud, etc., than a pump with the suction connection on the side of the tank, will appreciate what is meant by this amount which the pump will not take.

This water represents, in a tank of any ordinarily used dimensions, such a tonnage that one is sometimes led to wonder why the suction connection was not made at the bottom of the tank, under which conditions the tank itself could have been built several inches less in depth, or, conversely, with the same depth it would have held more available water. Fig. 167 shows a simple and proved arrangement which causes the pump to empty the tank down to the last half inch or less. An obvious modification would make it equally applicable in those cases where the suction connection is to the side of the tank.

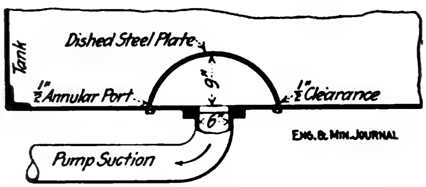


FIG. 167.—METHOD OF ATTACHING PUMP SUCTION TO BOTTOM OF A TANK.

The arrangement consists of a dished steel plate of about $\frac{3}{8}$ in. thickness, the overall diameter of which is about four times the diameter of the suction pipe. This plate is placed inside the tank with its concave side downward over the suction hole, and is distanced off the bottom of the tank about $\frac{1}{2}$ in. by four steel wedges, or by four setscrews tapped through the dished plate.

It is essential to have the dished steel plate heavy enough, or otherwise weighted, so that, when the pump, without first closing the suction or delivery valves, is stopped, the amount of water flowing back from the delivery lines will not be able to shift the plate. This dished plate being arranged by wedges or setscrews to clear the bottom of the tank by about $\frac{1}{2}$ in., will, in the case of a 6-in. pump with a plate 2 ft. in diameter, give as feed to the pump suction an annular port about $72 \times \frac{1}{2}$ in., consequently partly compensating the friction by excessive port area.

Slippage in Reciprocating Pumps.—Losses by slippage in reciprocating pumps, according to Mervin K. Baer, in *Power*, Apr. 8, 1913, is due to three causes: (1) Leakage past valves, both suction and discharge; (2) leakage past pistons or plungers; (3) entrained air and air pockets. The first is the most important and is due to the necessarily late seating of the valves, to the wearing of the valves and springs and to improper seating. The second is due to wear on piston or plungers and is usually greater with inside-packed plungers than with the outside-packed type. Entrained air is caused by low suction pressures; air pockets are due to faulty construction. Neither of these last two losses should be found in a well-designed pump. Slip reduces the amount of water pumped

and increases the power for a given quantity of water. The statement that it is generally about 5% is misleading, as it varies greatly. Piston speed and discharge pressure enter prominently into the question. Minimum slip occurs at the normal rated speed of the pump. It increases rapidly with a decrease in speed and increases slowly with an increase in speed. It is apparent that it is better to run a pump at high speed rather than low, so far as slippage is concerned. The slip also increases greatly with the pressure. Tests on pumps of the Chicago water-works system, under normal operating conditions, show a slip varying from 11.7 to 28.5%. A small, boiler-feed pump, running under normal conditions at about 75 lb. pressure, showed a slip of 58%. Slip varies greatly with the condition of the pump. Thus on a 40,000,000-gal. pump a slip of 1.7% was recorded on a certain date, the pump being in excellent condition. About 4 months later, this had increased to 28.5%. The necessity of considering slippage in pump calculations is evident.

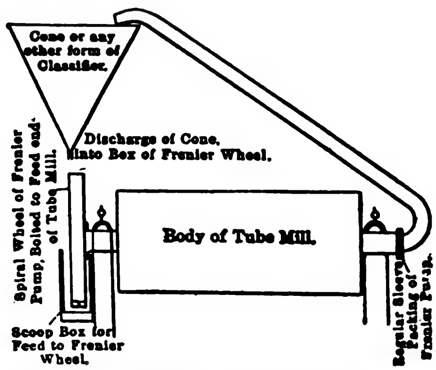


FIG. 168.—RETURNING PULP TO CLASSIFIER.

Method of Returning Pulp to Classifier from Tube Mill (By Cooper Shapley).—The difficulties of returning the pulp from the discharge end of a tube mill to the original classifier at the feed end are numerous. The principal machines used for this purpose are the bucket elevator, centrifugal pump, Frenier spiral pump and the air lift. Of these, the bucket elevator and the Frenier pump are the most used because of the slow speed and therefore slight amount of wear to their parts. But the objection to any of these forms of returns is found in the fact that, if for any reason, the power of the mill is shut off without warning, there is a heavy flow of pulp from the tube mill as soon as it stops that cannot be taken care of by any of the above returns, because they also are stopped. The scheme of returning the pulp as shown in Fig. 168, is a combination

of the tube mill itself with a Frenier pump, the spiral wheel of the pump is bolted to the feed end of the tube mill and the sleeve packing joint is bolted to the discharge end of the tube mill. A feed box is placed so that the discharge from the classifying cone drops directly into it, and the spiral wheel scoops it up and feeds it into the tube mill and out the discharge end, up into the original cone classifier. This pumping action of

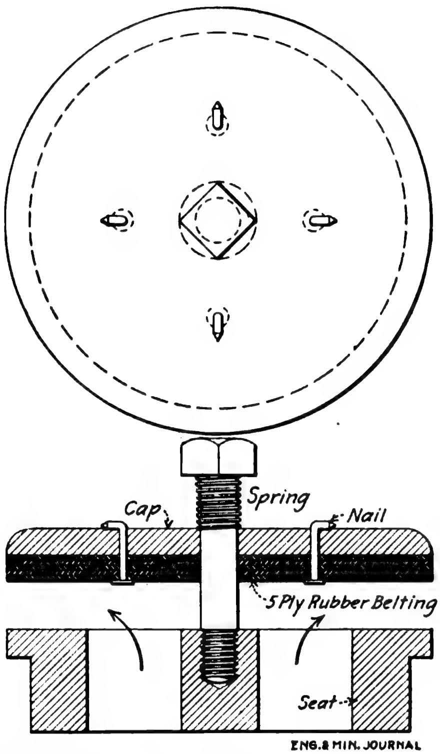


FIG. 169.—PUMP DISKS OF OLD RUBBER BELTING.

the spiral causes pressure inside the tube mill and therefore the mill is always a little more than half full of pulp, thus giving it a greater grinding capacity. If the power is shut off there is no slop or spilling of pulp to a sump. No extra motor is required to run the return machinery and the only wear is at the packing sleeve, which is no greater than in the Frenier pump. The speed of a 5-ft. tube mill corresponds closely to the speed of a 54-in. Frenier spiral.

The efficiency of a Frenier sand pump decreases rapidly as the height of the discharge pipe is increased. When two or more pumps are to be used in series, one discharging into the other next above, the height of

discharge pipe for the upper pumps should be slightly less than that of the lowest, then there will be little possibility of stoppage by the lowest pump supplying more material than the others can lift.

A Valve Protector (By Chester Steinem).—Lift valves, such as are used on the Deming and Gould pumps for pumping mill solutions, are usually subjected to severe wear, especially when operating on decanted solutions. A method of confining the wear to one easily replaceable part is shown in Fig. 169. At little expense ball valves may be altered so as to permit applying the same principle. A number of caps are kept on hand and these are pierced by four holes. Waste pieces of rubber belting are nailed and clinched to the caps and trimmed as shown. Punches, made of pipe ground to a cutting edge, would save time in punching. When a valve becomes worn, it is a simple matter to replace it.

V

NOTES ON THE EQUIPMENT OF METALLURGICAL PLANTS

TANKS AND ACCESSORIES

Volumes of Prismoidal Tanks.—For computing partial volumes of rectangular tanks or reservoirs having sloping sides, Douglas McLean, Coolgardie, Western Australia, recommends, according to *Engineering News*, the method of constant second differences. Dividing the tank into zones or layers of equal height, the difference between the contents of one zone and the contents of the zone next below is the first difference; the difference between two successive first differences is the second difference. The second difference is constant for a tank of the shape mentioned. To apply this fact, calculate the volume of three successive zones, find the first differences and the second difference, and add the latter successively to the first differences, thus obtaining all the first differences. The latter, added to the volumes that have already been calculated, will give all the zone volumes.

A little calculation will be saved by using the formula for the second difference:

$$D'' = 8mnh^3$$

in which h is the depth of the zone, m is the ratio $\dfrac{\text{horizontal}}{\text{vertical}}$ for the side slopes, and n is the corresponding ratio for the end slopes.

Example.—Rectangular reservoir with side and end slopes 1 on 2; bottom dimensions 286 by 135 ft.; depth 14 ft.; find the contents of zones 6 in. deep.

Here $m = n = 2$, and $h = \frac{1}{2}$, so that $D'' = 8 \times 4 \times \frac{1}{8} = 4$ cu. ft. Computing the volumes of the two lowest zones, we get 19,516.17 and 19,941.17, with a first difference of 425. Then the successive first differences are 425, 429, 433, 437, etc. The volumes are then found.

Zone	Second diff.	First diff.	Volumes
$0 - \frac{1}{2}$	4.00	...	19,516.17
$\frac{1}{2}-1$	constant	425	19,941.17
$1 - 1\frac{1}{2}$	constant	429	20,370.17
$1\frac{1}{2}-2$	constant	433	20,803.17
$2 - 2\frac{1}{2}$	constant	437	21,240.17
$2\frac{1}{2}-3$	constant	441	21,681.17

Precisely the same method of calculation applies to conical tanks, *i.e.*, circular tanks with sloping sides, except that the second-difference formula is slightly different. Calling m the side slope $\frac{\text{horizontal}}{\text{vertical}}$ and h the depth of zone, the formula is

$$D'' = 2\pi m^2h^3 = 6.283m^2h^3$$

which is constant.

Painting Cyanide Tanks.—Where wooden tanks are used in cyanide works it was formerly the custom to paint them both inside and out. For inside painting the well-known P & B paint was used almost exclusively, although in the Mexican plants a native asphaltic tar known as *chapopote* was used with good results. Due, however, to the fact that it was found quite impossible to prevent the cyanide solution from reaching the wood, the practice of inside painting was discontinued and it is now exceptional to find a wooden cyanide tank painted on the inside. In practice the tanks thus untreated are tighter than those which have been painted. In painted tanks, the solution was found to get through to the wood in some spots and not in others, resulting in unequal expansion of the wood and the opening of interstices. The outside of wooden tanks are almost always painted. If this is not done the alternate wetting and drying causes rotting which sets in at the outside and works in. The final result is a rotten stave or bottom plank through which the solution seeps and which cannot be remedied except by inserting a new piece of wood. There is no particular kind of paint required for the outside of the tanks, the only requisite being a paint of good body which will protect the wood. The solutions do not come in contact with this paint to any extent and therefore cannot be harmed. Iron tanks are usually painted on the outside as a preventive of rust, but are rarely painted on the inside. Graphitic or asphalting paints are good protectors against rust and are often used, although lead paints may be sometimes preferred.

Impervious Concrete Tanks. (By Alfred Moyer).—Concrete tanks may be rendered impervious, odorless and sanitary, provided the interior surfaces be treated as follows: After the forms have been removed, grind off with a carborundum stone any projections due to the concrete seeping through the points between the boards. Keep the surface damp for two weeks from the placing of the concrete. Wash the surface thoroughly and allow to dry. Mix a solution of one part soluble water glass (sodium silicate) 40 deg. B., with four to six parts water, total five to seven parts, according to the density of the concrete surface treated. The denser the surface the weaker should be the solution. Apply the water-glass solution with a brush. After 4 hr. and within 24 hr., wash off the surface with clean water. Again allow the surface to

dry. When dry apply another coat of the water-glass solution. After 4 hr. and within 24 hr., again wash off the surface with clear water and allow to dry. Repeat this process for three or four coats, which should be sufficient to close all the pores. The water glass which has penetrated the pores has come in contact with the alkalies in the cement and formed an insoluble hard material to a depth of $\frac{1}{8}$ to $\frac{1}{2}$ in., according to the density of the concrete. The excess sodium silicate which has

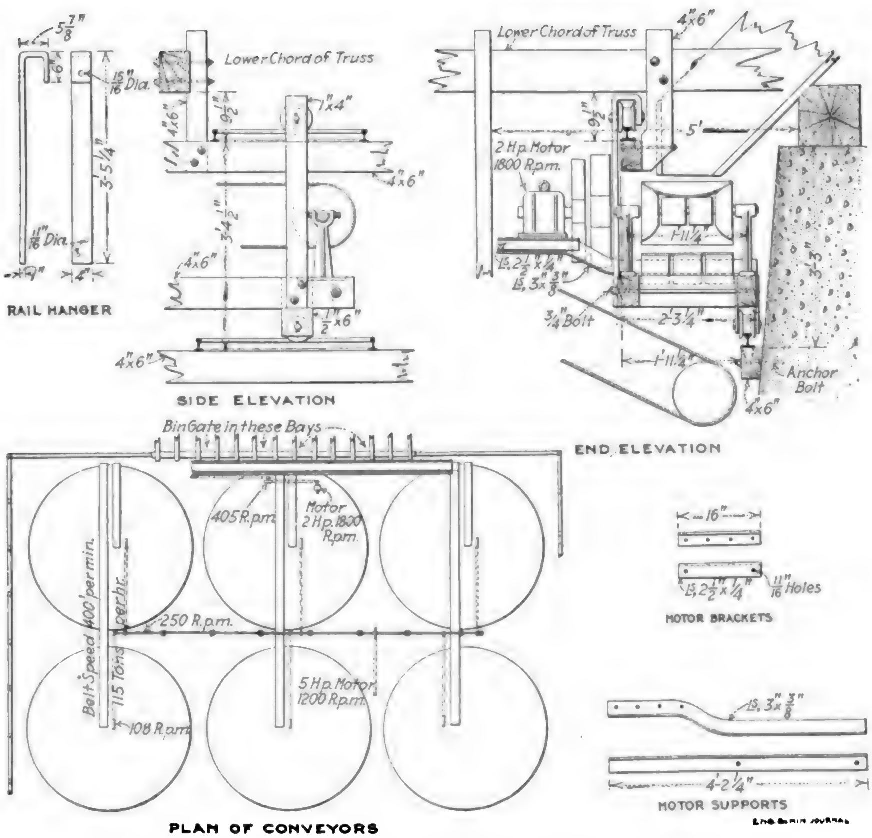


FIG. 170.—TANK-CHARGING SYSTEM AT WASP NO. 2 MILL.

remained on the surface not having come in contact with the alkalies is soluble, therefore easily washed off with water. The reason for washing off the surface between each coat and allowing the surface to dry, is to obtain a more thorough penetration of the sodium silicate.

Charging Tanks by Conveyors (By Jesse Simmons).—At the Wasp No. 2 mill, at Flatiron, S. D., belt conveyors are used to charge the dry crushed ore to the leaching vats, as is shown by the drawings in

Fig. 270. The ore is reduced to approximately $\frac{1}{8}$-in. particles by an equipment of two No. 6 and two No. 4 Gates crushers and four sets of 16 × 36-in. McFarlane rolls, the product passing to a 2 × 7-ft. screen set at an angle of 45 deg., from which the oversize is returned to the rolls. This screen has No. 6 wire, the opening being $\frac{1}{4} \times \frac{3}{4}$ in. The finished product drops to a bin of 700 tons capacity.

Parallel to the front of this bin is a conveyor, suspended on a carriage so that it can be moved in its entirety to discharge over its end to individual cross conveyors running to the tanks. Thus trips and sharp turns in the belt, which cause heavy wear on the belts, are eliminated. The ore is drawn through chutes from the bin, to the carriage conveyor, the latter being operated by a 2-hp. motor. This carriage moves on cast-iron wheels which operate on 12-lb. steel rails; the upper or outside rail is suspended on 4 × 6 timber brackets from the lower chord of the tank-room truss, and the lower or inside rail is spiked to a 4 × 6 which is bolted to the concrete retaining wall under the front of the finished-ore bin. The carriage is constructed of 4 × 6-in. timbers, strengthened with angle iron and plates. Troughing pulleys are spaced 3 ft. 11 in. between centers and idlers 5 ft. 8 in. The length over all of this conveyor is 51 ft. The conveyors to the tanks, which are six in number, or one for each tank, are operated from a line shaft driven by a 5-hp. motor. This set of conveyors rests upon ties laid on the lower chords of tank-room roof trusses, which gives the top of the belts an elevation of 9 ft. above the top of the tanks at point of discharge. The tanks are 32 ft. diameter by 9 ft. deep. A little shoveling is required to finish the charging of a tank, as the crushed ore when discharged from the conveyors piles up in a cone having a 45 deg. slope. The mill handles 500 tons per day, or to be exact, 12 tanks in 10 days. The tanks have a capacity of 420 tons each. All of the conveyors have 18-in. belts, operated at 400 ft. per min., giving a capacity of 115 tons per hour.

Discharge Doors for Cyanide Tanks.—All tanks in a cyanide plant, except those to contain solution returned from precipitation sumps, should be provided with bottom discharge doors of generous dimensions when built. Sooner or later it will be necessary to clean out such tanks. With slime settlers, agitation tanks, battery and supply tanks, nothing is so convenient as bottom discharges, nor so expensive as raising slime over the edge of the tank.

Gate for Leaching Vats.—The North Star mills, at Grass Valley, Calif., are using a discharge gate for leaching vats that has proved satisfactory in operation and cost of maintenance. Referring to Fig. 171, M is an annular casting bolted to the floor of the vat. Provision is made for calking filter cloth tightly. A tripod P, cast integral with M, serves

as a guide for the valve-stem and a support for the operating mechanism. The lower face of *M* is machined and forms the valve seat. A groove is cut in the valve seat and a piece of round rubber packing *D* is fitted in place as a gasket.

The valve gate *V* is held in place on the stem by a nut and packed against leakage along the stem by rubber gasket *E*, and cut washer *F*. The valve stem *A* of sufficient length to reach above the surface of the charge, is made from ⅞-in. square wrought-iron bar, and is threaded at its upper end. It passes through a square opening, in the hub of the tripod, the upper portion of which is enlarged and threaded to take a standard pipe of 1½-in. diameter. A piece of pipe somewhat shorter than the valve stem is screwed into the tripod, and capped by a reducer at its upper end. This reducer *C* serves as a bearing for the operating hand-wheel *W*, running on the thread *A* of the stem. This gate is moderate in first cost and can be made in any foundry. It opens away from the

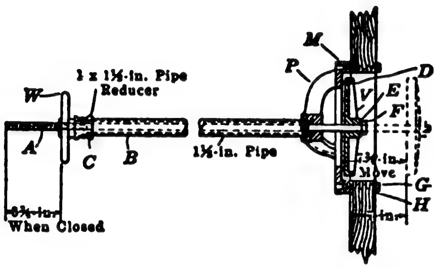


FIG. 171.—OLIVER SAND-VAT DISCHARGE GATE.

charge, the gate dropping 7¾ in. below the valve seat, which has an internal diameter of 10 in., thus providing ample discharge area. All troublesome guy wires are eliminated and the operating mechanism is above the charge. This gate was designed by E. L. Oliver, and is not patented.

Discharge Gate for Leaching Vats.—The construction of a discharge gate for cyanide-plant leaching vats that was designed by Philip Argall as early as 1906 is shown in Fig. 172. The characteristic of the gate is that the valve is dropped from its seat by gravity to full open position. The valve as shown in the sketch is designed for use in steel vats. The tripod *A* is self supporting and at its upper part is threaded to receive a 3-in. pipe *B*. This pipe extends above the surface of the charge in the vat and terminates in a special casting *C* threaded to receive it. Upon the upper face of this casting, which is machine finished, rests a circular notched plate *D* bolted at one side to the casting *C*. Through the pipe *B* there extends the valve rod *E* bolted to the valve at the bottom and terminating in a special head *F* to the eye-bolt in which is attached a

chain. This special head is threaded to the valve rod. To open the discharge valve the head *F* is given one turn which loosens the grip on *D* enough to permit slipping that plate to one side so that the head *F*, the base of which is larger than the notch in *D*, can slip down with the valve rod through the pipe *B*. The drop of the valve is limited by the bolts below the bottom of the vat. In closing the gate the valve is pulled up by

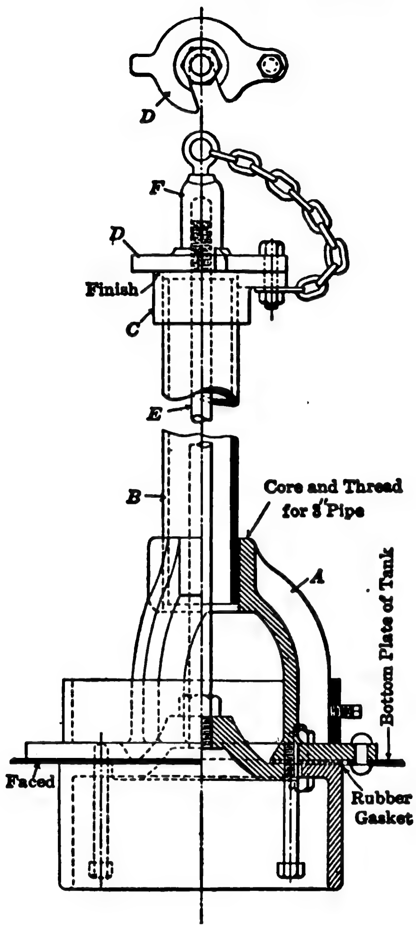


FIG. 172.—THE ARGALL DISCHARGE GATE FOR LEACHING VATS.

the chain until the head *F* is raised to such position that disk *D* may be slid in place beneath it and the valve is then brought to bear on its seat by screwing *F* tightly down upon *D*, the disk *F* being enlarged at the bottom to give ample bearing area around the notch in the disk. This gate resembles the Oliver gate illustrated in Fig. 171 but differs in the method of operation.

A Flexible Decanting Hose (By W. P. Lass).—The concentrate from the Frue vanners in the Alaska-Treadwell and affiliated mills is cyanided after regrinding in tube mills at the new cyanide plant. Agitation in cyanide solution is effected in Pachuca tanks in which the concentrate is allowed to settle at the end of the operation and the clear supernatant solution is decanted. Decantation takes place (*Bull.* A. I. M. E., February, 1912) through a flexible hose which is made by wrapping canvas coated with tar around pieces of old boiler-tubing, 3 in. in diameter and 4 in. long, spaced ¾ in. apart. The canvas between the short lengths of tubing is wrapped with wire, making the diameter of these spaces slightly smaller than that of the tubing, thus insuring flexibility as well as avoiding the shifting of the tubing. Attached horizontally to the top

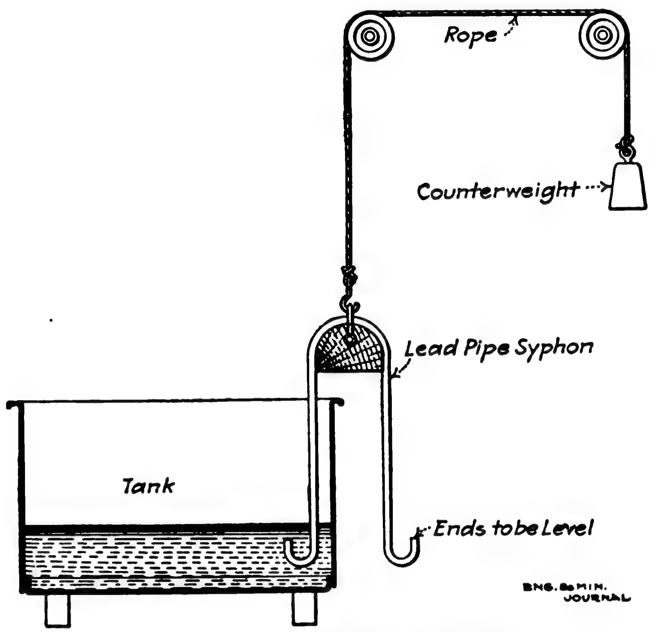


FIG. 173.—AN EVER-READY SIPHON.

of the flexible hose is a 3-in. slotted pipe. In operation this slotted intake floats by the aid of two adjustable air-cylinders. The arrangement of these cylinders permits the submergence of the intake pipe to any depth desired.

An Ever-ready Siphon.—A contrivance suitable for siphoning clear liquid from an acid-treatment tank, or for similar uses, is described by P. T. Morrisby (*Journ.*, Chem., Met. and Min. Soc. of So. Afr., February, 1913). It consists of a pipe of iron or lead, preferably the latter, long enough to reach one leg to the bottom of the tank. This is bent in the

middle over a semicircular block of wood, and the two ends of the pipe are turned up, as shown in Fig. 173. A rope and counterweight allows convenient movement. After the original priming of the device, under pressure, it remains primed and is always ready for use. Beside its application in connection with the acid-treatment tank, it is serviceable for other minor operations about the plant, such as removing clear liquor from settled precipitate, emptying zinc-box compartments, and other similar uses.

Sluicing Out Sand Tanks (By Claude T. Rice).—At the North Star company's cyanide plants at Grass Valley, Calif., a clever way of sluicing the sands out of the sand tanks is employed. A somewhat similar system is used at the Sierra Buttes plant, but when the method was worked out for the North Star some modifications were made. The device consists of a built-over Butters-Mein distributor on the distributor box of which a cover is bolted so water under pressure can be forced through the orifices on the rotating arms. The cover plate for the distributor box is made of cast iron and is bolted by cap screws to the distributor box at the point where the cone is riveted on the regular distributor. This cover plate has a stuffing-box arrangement for making a tight fit where the feed pipe passes through it. By means of this feed pipe the whole contrivance is hung from a carriage that travels along an overhead track, just as in the case of the distributor. The feed pipe, which is 3 in. in diameter, is welded to a 2 in. shaft that in turn carries the seat on which travel the balls that carry the whole sluicer.

Leading from the sluicer box are eight arms varying in length and having sluicing pipes fitted to them by means of T-joints. These sluicing pipes are at right angles to their respective arms, but are pointed at a slight angle from the vertical, not only to cause the pipe to rotate, but also to give a side motion to the sand which the jet of water stirs into, suspension. These arms end in orifices, $2\frac{1}{4}$ in. long and $\frac{5}{16}$ in. wide, while one of the pipes has an elbow at the end from which water is also discharged. This elbow can be turned so as to discharge ahead, or toward the rear, and in this way the speed at which the sluicer revolves is regulated.

The sand tanks are 22 ft. in diameter and have a depth of 87 in. These are filled until they hold 100 tons of sand. A sluice head of 20-lb. pressure is used, and it takes about an hour and a half to wash the sands out of a tank. In the tanks at the Central mill the last 3 in. are washed out by means of a hose, as the sluicer only tends to drive the sands around and around in the tank because of the very flat slope that the last of the sand assumes toward the center of the tank. This difficulty of discharging the last of the sands has been partly remedied at the North Star mill by placing one of the four discharge doors near the

side of the tank. As the sands flow around the tank toward the end of the sluicing operation, they are washed out of this door. Considerable water is required to wash the sands out in this way, but it is a cheap method where there is a good supply of water available. The sluicer is pushed around from tank to tank on the same track as the distributor. The upkeep is about the same as on a distributor, being practically nothing. The two in conjunction afford an efficient method of handling the sands at a cyanide plant.

Automatic Water Cut-off.—In many milling operations it is necessary to keep a constant head of water. This may be accomplished by means of the apparatus shown in Fig. 174. The one here described is used in connection with a Hancock jig. The valve *a* in the water pipe is operated

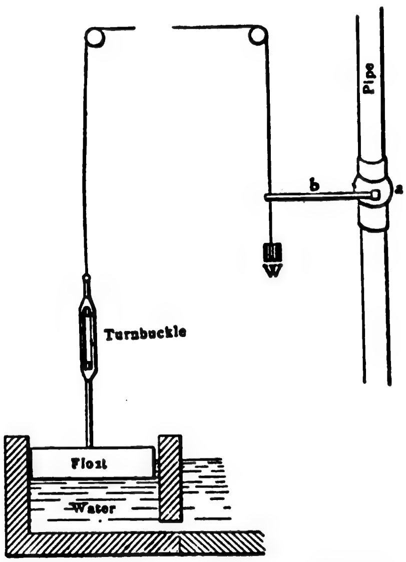


FIG. 174.—AUTOMATIC WATER CUT-OFF.

by an arm *b* about 20 in. long. To this lever is attached a cord which runs over pulleys to a float where the water is to be kept at a constant head. The rope is adjusted to the proper length as nearly as possible, and the final adjustment is made by means of a turn-buckle near the float. Also attached to the arm *b* is a weight *w*. As the water lifts the float the weight *w* pulls the arm down and this closes the valve. When the flow of water is too small the float sinks and the arm *b* opens the valve. The control of the flow of the water is entirely automatic.

Economical Tank Connection.—A simple arrangement of piping and fittings for a tank which automatically prevents overflow, and at the same time uses only a single pipe for both supply and discharge is illustrated in

Fig. 175. Referring to the larger drawing, the pipe *A* extending into the tank carries a balanced lever valve *B* operated by the float *C* in the usual manner. Obviously with the float valve alone, *A* could not be used for discharge after the float has closed the valve unless other means were used to open it. To overcome this difficulty, writes W. S. Mayers in *Power*, Feb. 25, 1913, the tee *D* is inserted and a bypass run into the tank, as shown. In the bypass there is a check valve *E* so placed that the water can flow from the tank, but not in the reverse direction. When the water reaches its highest level in the tank both valves will be closed. If

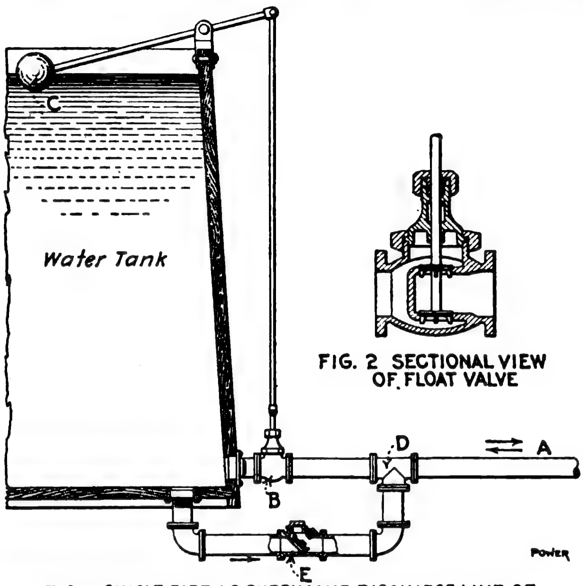


FIG. 2 SECTIONAL VIEW OF FLOAT VALVE

FIG. 1 SINGLE PIPE AS SUPPLY AND DISCHARGE LINE OF WATER TANK

FIG. 175.—ECONOMICAL TANK CONNECTION.

now the source of supply is cut off and an outlet from the pipeline opened the pressure will fall at the discharge end of the check valve, and the water will flow freely through the bypass as well as through the float valve when the float has dropped sufficiently to open it. In the smaller drawing is shown a sectional view of the float valve.

Float for Solution Sump Indicator.—Generally speaking, solution-sump indicators are troublesome for the reason that the cork or wooden floats usually get waterlogged and so become unreliable. A simple float that avoids this difficulty is described by Percy T. Morrisby (*Journ.* Chem., Met. and Min. Soc. of So. Afr., August, 1912). The float is made of a fair-sized drum, a 10-gal. paint drum serving the purpose admirably. The ends of the drum are made tight; two holes are drilled in the middle of the sides to receive a length of $\frac{3}{4}$-in. iron pipe, which is soldered in position at

right angles to the long axis of the drum. The object of the short piece of pipe is to hold the float in position; it should project about 1 in. on either side of the drum. A rod of ⅝-in. round iron, long enough to reach from the bottom to the top of the vat, serves as a guide for the float; this is screwed into a piece of flat-iron plate, say 12 × 12 × ½ in., which rests on the bottom of the vat about 9 in. from the side, the upper end of the rod being supported in a suitable manner. The float is placed in position on the rod and connected to the indicator board by pliable steel wire. To keep these drums in good order, a coat of tar applied twice a year is recommended.

Electrical Tank Signal (By Chester Steinem).—The electrical connections for indicating the depth of solution in a tank, by means of two signal

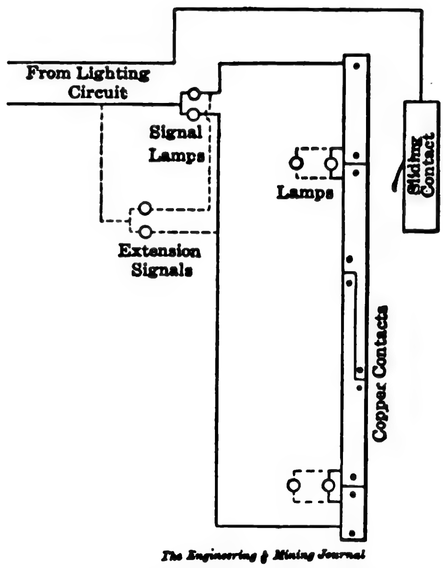


FIG. 176.—WIRING FOR TANK INDICATOR.

lamps, situated where desired, are shown in Fig. 176. The device was introduced at the Socorro mill, Mogollon, N. M., by D. J. Kennelly. The sliding contact is attached to a float in the tank by a cord over a pulley (not shown). If a four-strand block is used, the distance traveled by the sliding contact will be shortened enough to permit housing the whole. The dotted lines show an extra set of signals in case two places are to be provided with them. Here the contacts are so dimensioned that when the upper lamp burns at full candle power, the depth of solution is 15 to 15½ ft.; at half power, 13 to 15 ft. When both lamps burn at half power, the depth is 11 to 13 ft.; when the lower lamp burns at half power, 9 to 11

ft., and when the lower lamp burns full, the depth is 7 to 9 ft.; no light shows for depths less than 7 feet.

Electrical Reactance Water Level Indicator.—An extremely simple device for indicating water level in an unseen tank or reservoir is described by F. J. Storey in *Power*, Feb. 18, 1913. It is a reactance coil having a movable core, actuated by a float, as illustrated in Fig. 177, the water level being gaged by the brightness of the lamp, and is used to indicate the approximate height of the water in the forebay of a hydro-electric plant. Two 8-cp. lamps connected in parallel and placed side by side in a conspicuous position in the dynamo room were connected through 1800 ft. of No. 10 single copper wire to the reactance coil placed at the forebay up on the mountain side. The steel water pipe was used for the return circuit. The lamps burn brightly at low water and dim at high water and after a little experience the operator is able to judge very closely the intermediate

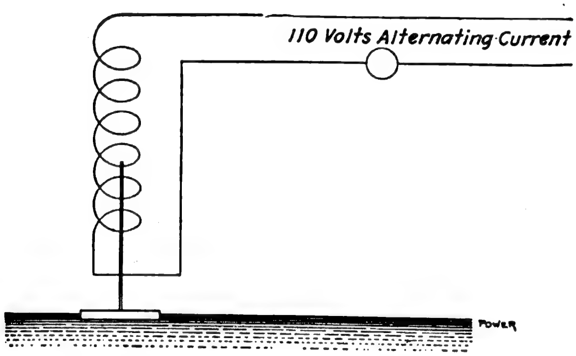


FIG. 177.—REACTANCE COIL IN CIRCUIT OF SIGNAL LIGHT.

points. The difference in water level to be indicated was 20 in. The coil was 20 in. long, made by winding 13 layers of No. 18 cotton-covered copper wire on a piece of fiber tube of $\frac{7}{8}$ in. inside diameter; core of $\frac{5}{8}$-in. solid soft iron, 20 in. long or over. One 16 cp. or two 8 cp. lamps were connected in parallel and put in series with the coil. The supply was 110-volt alternating current.

A Solution Meter (By Chester Steinem).—In many places in a cyanide mill, it is desirable to know the amount of solution flowing through a pipe, for instance, to the zinc boxes. Or, in a positive-pressure filter, like the Burt, where the leaves are inaccessible during a cycle, it is desirable to know the thickness of cake and the amount of wash going through. Knowing the percentage of dry slime in the pulp and the area of the filtering surface by measuring the effluent, the cake may be determined.

Measuring solution by flow over weirs, pump capacities, etc., is known to be inaccurate especially where large variations of flow occur. While

theoretically the apparatus here described, and illustrated in Fig. 178, is subject to error from the same source, practically it has been found to be surprisingly accurate under such conditions. It was designed for measuring feed water at Cananea.

Essentially, the meter is the usual sloping-bottomed tray, resting upon a "knife edge" and halved by a partition. The solution flows alternately into one compartment and then the other. The weight of the solution in the one compartment forces that side down, bringing the other compartment under the feed pipe. To the tray an upright lever is fixed, the free end of which rests in a slotted piston rod, the piston rod being attached to dash pots. A register or revolution counter is also attached to

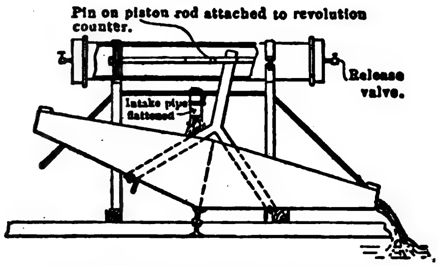


FIG. 178.—SIDE ELEVATION OF SOLUTION METER.

the piston rod, recording the number of oscillations of the tray. The machine should be calibrated for the conditions under which it will be used. This may be easily done by finding the number of counts of the meter necessary to fill a tank of known capacity and taking the average. The dash-pot cylinder is simply a closed pipe with oil holes and a release valve at each end to regulate the escape of air. The meter may be placed in a tank or simply boxed. A meter used for measuring the effluent of the Burt filters at the Socorro Mines mill, Mogollon, N. M., handles 600 tons in 24 hr. Each double oscillation (meaning both compartments filled once) equals 2.83 cu. ft. or 176⅞ lb. of solution.

The principle of the device is not new, except perhaps for the dash pots; they offer a resistance to the tilting tray so that a larger volume of solution is measured at each turn, the operation is steadier and the change of position is made without a jerky motion. A meter of this type can be used for measuring tailings and concentrates where only a rough approximate weight is desired and when the proportion of water to solids is known.

Solution Meter at the Belmont Mill (By C. S. McKenzie).—The accompanying drawings, Fig. 179, show a tilt box in use at the Belmont mill, Tonopah, Nev., for measuring solution. The box is suspended on timbers over the tank into which it discharges, and is fed from a 6-in. pipe. In the drawing to the right the meter is shown in the position it

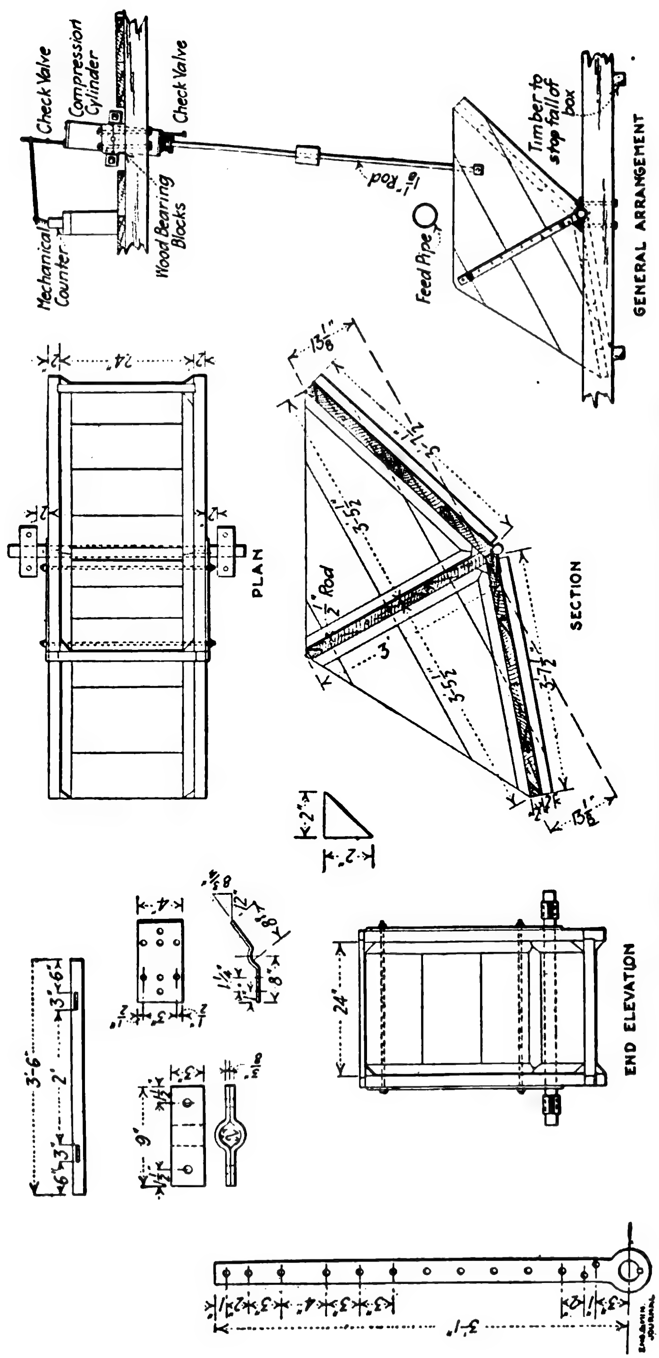


FIG. 179.—DETAILS OF SOLUTION METER, BELMONT MILL.

assumes when the right side is filling. As this side becomes full, the weight is sufficient to overbalance the left side, and the box tips to the right, pulling down the piston rod and discharging into the tank. As the piston is pulled down, it pulls the piston in the compression cylinder down, tending to form a vacuum in the upper half of the cylinder, and compressing the air in the lower half. This compression retards the falling box sufficiently to let it down gradually and prevent jarring. Air is drawn through the upper check valve into the upper half of the cylinder as the piston goes down. This air is compressed as the piston is pushed up by the box filling and tipping to the left again, and more air is drawn through the bottom check valve into the lower part of the cylinder, etc. To prevent the compression in the cylinder from becoming great enough toward the end of the stroke to stop the fall of the box before the latter has completely discharged, the check valves are allowed to leak enough air to make the rate of fall uniform. The amount of compression can be regulated by the amount of air admitted through the check valves. The compression cylinder is supported at the middle, and the ends travel through arcs as the box tips. A mechanical counter is attached by a rod to the top of the cylinder, and registers each tip. The apparatus was calibrated by counting the tips required to fill the tank beneath, and was adjusted so that it tipped when just full by raising or lowering the timbers which stop its fall. The box is made of 2-in. Oregon pine lumber, and mounted on a piece of $1\frac{15}{16}$-in. shafting. The compression cylinder is an old automobile cylinder, but could be made of a piece of pipe or casing. About 196 gal. of 1.2 sp. gr. solution are handled per min., and after adjusting the intake of air to the cylinder, the arrangement works smoothly and without attention.

POWER TRANSMISSION DATA

Shafting and Belting Calculations.—The accompanying charts represent a simple graphical solution of the problems which arise concerning shafting and belting. While presenting nothing new in theory, the application will be found useful, says Miles Sampson (*Power*, Oct. 8, 1912).

The chart for shafting, Fig. 180, is based upon the commonly used formula for the strength of shafting:

$$Horsepower = \frac{(diameter\ of\ shaft)^3 \times r.p.m.}{constant}$$

In explanation of the constant, it may be noted that 100 is used for head-ends of shafts receiving large amounts of power. By largely increasing the size of shaft necessary to carry a given power, it takes account of the large transverse stress due to the belt pull, and of the nec-

essary stiffness. Any shaft carrying pulleys with belts over 6 to 8 in. wide should be figured using this constant. The constant 50 should be used to calculate countershafts driving ordinary machinery, and shafting carrying small belts. When 30 is used only the transmission of power should be expected of the shaft, no transverse stresses being considered.

The method of using the chart follows: On the scale representing the given speed follow vertically to the line denoting the proper constant, then across horizontally to the line representing the shafting size. The abscissa of this point on the horse-power scale gives the allowable load. Thus, at 300 r.p.m. a $3\frac{1}{8}$-in. shaft carrying small belting (constant 50) would transmit 180 hp. Problems having different given conditions may be solved in a similar manner. For example—to carry 400 hp. at 400 r.p.m. a head-end shaft must be $4\frac{1}{2}$ in. diameter.

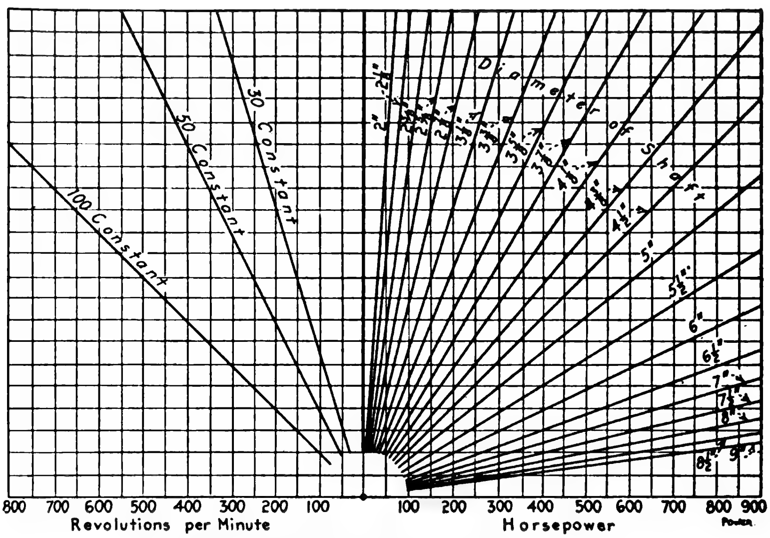


FIG. 180.—CHART FOR CALCULATING SHAFTING.

The chart for belting, Fig. 181, is based upon a speed for double belts of 600 ft. per min. per horsepower per in. of width. This is conservative in the light of present practice, but represents a standard toward which belt users are tending. For wide belts it may doubtless be decreased to 500 ft. per min. with perfect safety. For single belts the 600 becomes 1200, and for triple belts, 400.

For using the chart at any number of revolutions per min. follow vertically to the line representing the pulley diameter, which gives, on the vertical scale, the belt speed in feet per min. Follow across horizontally to the belt width and down to the required horsepower. Thus at 400 r.p.m. on a 36-in pulley (3770 ft. per min.) a 20-in. double belt

will transmit 126 hp. Similarly, if a 30-in. belt is the maximum width available for transmitting 250-hp., a belt speed of 5000 ft. per min. is necessary; this requires a 60-in. pulley running 320 revolutions per minute.

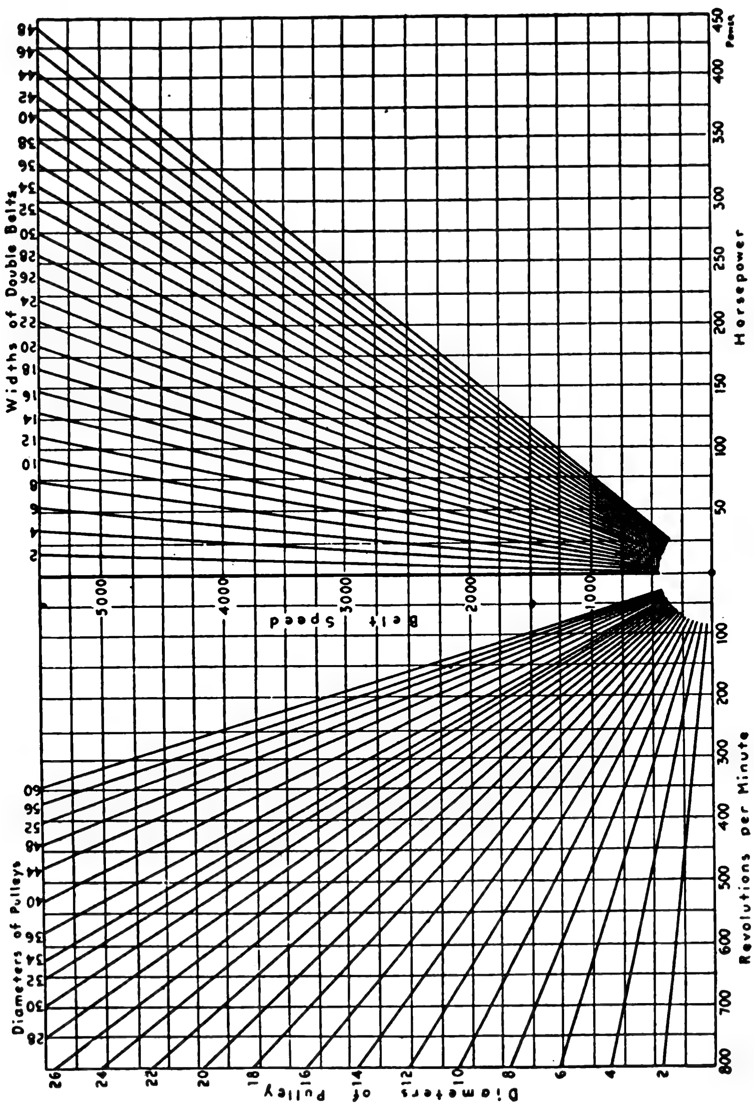


FIG. 181.—CHART FOR DOUBLE BELTING.

Power Consumption in Bearings.—A series of tests was made in factories driven by electric motors to determine the power consumed in bearings. The power required to drive the motors with different loadings

could be accurately measured and by subtraction, the power absorbed by the line shafting obtained. The results are given by C. A. Graves, under whose direction the tests were made (*Amer. Mach.*, June 5, 1913).

In obtaining the data, ammeters and voltmeters were used to measure the power of direct-current motors, and watermeter for alternating-current motors. The testers were instructed to count the hangers in which the shafting revolved, and also the loose pulleys on the shafting and countershafting over which the belts were passing. Then all work was stopped on the various machines and the amount of power input into the motor was measured. All the belts were then removed from the shafting and countershafting, one at a time, and readings taken of the amount of power input into the motor after each belt was removed, until only the motor driving-belt remained. The motor driving-belt was then removed, and the amount of power required to drive the motor free measured. In many instances, the belts were replaced in reverse order to check the results.

The difference between the amount of power required with the belts on and the amount required where the motor was running free, represents the amount of power absorbed by the bearings, subject to a slight correction due to the efficiency of the motor at various loads. Under these conditions of test, however, the extra tension on one side of the belt when operating a machine, and consequently the increased friction on the bearing, is not measured. This amounts to approximately 20%, but as this increased friction is due to work done, and occurs only when the machine is operating, it is not considered in this article.

Referring to the classifying of a hanger and a loose pulley together as a bearing, it was found that the average loose pulley, either on a shaft or countershaft, absorbed nearly as much power as a hanger, and in some cases it was found that nearly twice as much power was used by a loose pulley as an adjacent bearing on the same shaft. Therefore, as the tests progressed, because there was so slight a difference between these two, both were classed as bearings, and they were counted together.

The results of these tests show that the average shafting in the stone-working industries uses the largest amount of power and this is due to lack of care in keeping the shafting in alignment and also to dusty, gritty surroundings. In this line of business, on the average, a horse-power would be absorbed by approximately five bearings. In machine-shop practice, cases of excessive friction were found which were due to lack of lubrication, but in the average shop the shafting losses per bearing were not excessive, although in a great many cases the total loss because of excessive bearings was more than necessary. In the various industries the average bearings absorb approximately 0.12 hp. each. This means that 100 bearings will use 12 hp. in friction during every hour of operation.

Few realize what a saving can be effected by a little attention given to this subject; for instance, the operation of an extra length or two of shafting just because it might be used some time, may mean a waste of 1000-hp.-hr. per year.

The data were classified in order to observe the effect of the size of the shafting, as shown in Table XXXII. The smaller fraction diameters are classified under the nearest simple fraction. For example, 2 in. diameter includes sizes between 1$\frac{15}{16}$ in. and 2$\frac{1}{8}$ inch.

TABLE XXXII.—BEARING FRICTION OF VARIOUS SIZES OF LINE SHAFTING

No. of bearings	Size of shaft	Average r.p.m.	Maximum hp. per bearing	Minimum hp. per bearing	Average hp. per bearing
66	$\frac{15}{16}$ in.-1	428	0.052	0.010	0.036
706	1$\frac{1}{8}$ in.	382	0.079	0.016	0.033
37	1$\frac{1}{4}$ in.	425	0.119	0.040	0.062
492	1$\frac{1}{2}$ in.	392	0.193	0.035	0.089
155	1$\frac{3}{4}$ in.	218	0.113	0.029	0.078
409	2 in.	242	0.300	0.028	0.133
21	2$\frac{1}{4}$ in.	264	0.321	0.124	0.257
83	2$\frac{1}{2}$ in.	243	0.300	0.085	0.255

A test was made in a factory with a complete equipment of Hyatt roller bearings and the result of the test on the eight separate shafts is given in detail in Table XXXIII. The average friction loss of these 216 bearings amounts to 0.0286 hp. By comparing this loss with the losses of 2-in. babbitted bearings it will be noted that there is a reduction of approximately $\frac{1}{10}$ hp. per bearing by the use of roller bearings. What the reduction in power will save in dollars and cents, however, must be taken up separately for each installation.

TABLE XXXIII.—FRICTION OF HYATT ROLLER LINE SHAFT BEARINGS

Section of shaft.....	1	2	3	4	5	6	7	8
Number of hangers.	22	7	11	7	7	7	7	8
Number of loose pulleys.............	40	24	20	19	5	8	2	22
Total bearings......	62	31	31	26	12	15	9	30
Horsepower to drive shafting..........	1.716	0.858	0.724	0.804	0.288	0.549	0.281	0.938
Horsepower for bearing..............	0.027	0.028	0.026	0.031	0.024	0.036	0.031	0.031
R.p.m. shafting.....	275	300	300	200	275	200	240	180
Diameter shafting...	2 in.	2 in.	2 in.	1$\frac{15}{16}$ in.	1$\frac{15}{16}$ in.	1$\frac{15}{16}$ in.	1$\frac{15}{16}$ in.	1$\frac{15}{16}$ in.

Reverse Belt Drives.—The question of getting a reverse motion with a belt drive without crossing the belt has recently been discussed in *Power*. The two methods shown in Fig. 182 were given as solutions

of the problems, but the upper method is supposed to be preferable, because in this the pull is entirely from driver to driven wheel, and the only pull on the tightener and idler is that due to their friction and the weight of the belt. In the second case there is the direct pull of the whole drive against the idler. It must be noted, however, in laying out a drive as shown in the upper illustration, that if the driver and driven wheels are not of the same size, the idler and tightener must be situated near the smaller of the two.

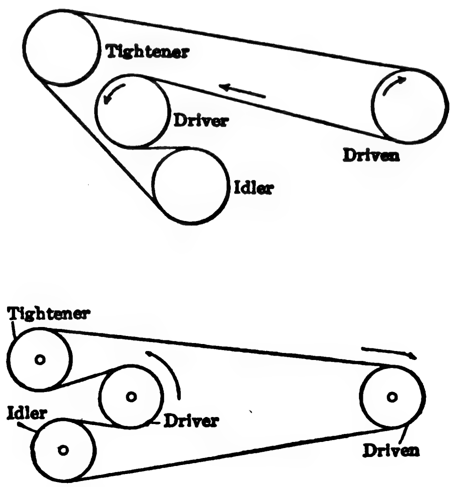


FIG. 182.—ARRANGEMENT OF PULLEYS FOR REVERSE DRIVE.

Quarter-turn Belts.—Quarter-turn belts are often a source of much annoyance, trouble and delay. This, when caused by the unequal stretching of belt edges, can be neatly and permanently avoided by putting a half turn in the belt before lacing it, as shown in Fig. 183. When run this way both sides of the belts are alternately on the pulleys and both edges alternately take the stretch of the quarter turn. Theoretically, the hair side of a leather belt alone should touch the pulley, but the extra strain of the quarter turn, especially with wide belts, large pulleys or short centers, warrants this departure from standard practice. With rubber belts the advantage of this half turn is particularly noticeable and the life of the belt is prolonged and that of the operator is made more pleasant.

Care of Rubber Belts.—Methods of handling and splicing rubber belts are of interest to the mining industry. Some valuable kinks are given by Robert Moore in *Power*, July 22, 1913. Care must be taken in putting on a new belt to stretch it as much as possible before splicing. In placing a 36-in. eight-ply rubber belt on pulleys 50 ft. between centers,

allow ⅛ in. per running foot for stretch. Put on the clamps, as shown in Fig. 2, referring to Fig. 184, and draw them tight. Do not be afraid of breaking the belt, as even a five-ply 10-in. belt will stand a strain of 10,000 lb. and larger ones in proportion; the pulley will collapse first. Take all the tension the bearings will stand, then turn the shaft slowly back and forth until the clamps touch the pulleys; taking up the slack as it is recovered from the upper half. Neglect to do this will stretch only one-half of the belt, and is apt to cause it to run out of line.

After thoroughly stretching, proceed with the lap, which in this case should be 45 in. long. It should always point in the direction of travel over the pulley, as in Fig. 1. Thus any slip of the pulley will have a tendency to smooth down the lap. Place a board on the clamp rods on which to rest the splice and draw a line squarely across the belt 47 in.

FIG. 183.—QUARTER-TURN BELT, LACED WITH HALF TWIST.

from the end. Lay the section off in 2-in. squares, starting 1 in. from the outer edge, as in Fig. 2. Punch ¼-in. holes where the lines cross. As there are eight plies, there will be three cuts or scarfs. Cut a line just the depth of two plies at the 45-in. line and peel off these two thicknesses; do the same at the 15- and 30-in. lines.

Scarf the other end, place the halves together and punch holes in the lower half, inserting the punch in the holes in the upper half. By so doing, the holes will be directly opposite each other. Cleanse the surfaces with naphtha and apply a liberal coating of the best rubber cement. Allow this to dry until it will not stick to the fingers, then place the laps together, starting at the edge and rolling the upper one out as it is being cemented, so that air may not be entrapped between the surfaces. Sew the outer edges, using the shoemaker's stitch, shown

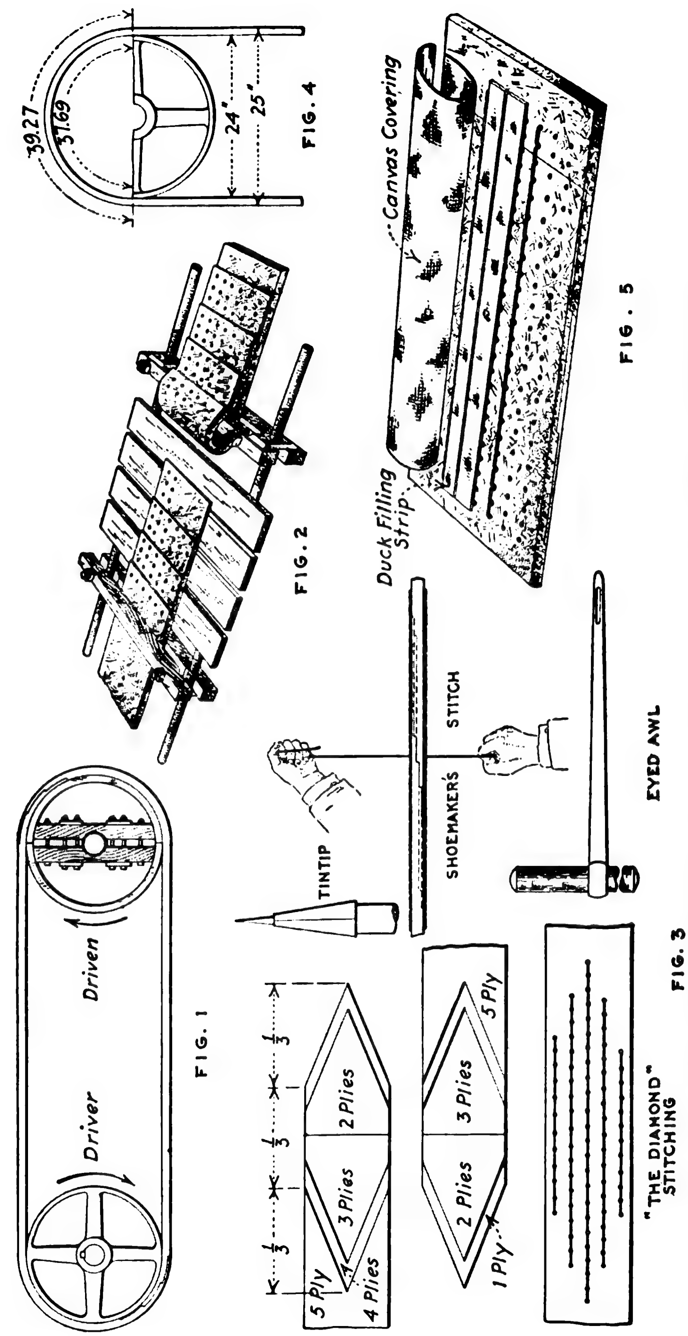


FIG. 184—STITCHING, SPLICES AND TOOLS USED IN REPAIRING RUBBER BELTS.

in Fig. 3. Alternately roll and pound the joint until it is perfectly flat, then sew each row of holes as was done with the outer ones.

Cut filling strips of duck the width of the space between lacings (most belt companies sell this duck all prepared) and give them several coats of rubber cement on each side. Clean the face of the belt between the laces with naphtha and give it a liberal coating of cement. When each surface is dry, that is, when the finger placed lightly to the surface will not adhere, place the strips between the lacing and roll them down. Now take a piece of duck, the width of the splice, but 4 in. longer, and cement this to the face of the belt, covering the joint. By so doing the lacings are protected and, except for the occasional renewal of the outer covering the joint is as durable as the belt itself. The manner of this is shown in Fig. 5. For sewing, tip the laces like a shoe string by bending a V-shaped piece of tin around the ends, as illustrated, or use the eyed awl shown, to pull the lace through.

Belts smaller than 10 in. are butted and have a butt strap on the outer surface; this is known as the "back splice." Belts on grindstones, saws, rattlers, etc., where shippers are used will wear on the corners where they are butted, and the outer lace hole will soon tear away, if this form of joint is not used. Shippers for belts of this kind should always be of the roller type. In joining canvas belts always stagger the holes and do not have them less than 1¼ in. apart as the fabric is apt to crack across in cold weather.

Another form of joint, known as the "diamond splice," is used on generators with small pulleys, as there is less shock when the lap passes over the pulley, eliminating all flicker of the lamps. The diamond splice is made in much the same manner as the lap splice except that the scarfs are divided into three equal parts and cut as shown in Fig. 3; it is not so strong a joint as the lap splice, but is more flexible and, therefore, better adapted to small pulleys.

Increasing the diameter of the pulley lengthens the life of a belt, as the stretch and compression are less per foot. As the thickness of the belt increases, so should the diameter of the pulley increase. For instance, the arc of a belt ¼ in. thick on a 2-ft. pulley would measure 37.69 in. next the pulley face, while the outer edge would measure 38.48 in., a total distortion of 0.79 in. If the belt were ½ in. thick the outer edge would measure 39.27 in., giving a total distortion of 1.58, just twice as much (Fig. 4). When more power is wanted and it is not advisable to increase the size of the pulley, better results are obtained by widening the belt and pulley face than by increasing the thickness of the belt. To increase the diameter of the pulley to 48 in. would give a total distortion of 1.58 in. when using a ½-in. belt, but as the arc would measure 75.4 in., the distortion for each lineal foot would be halved.

Animal fats and grease should never be used on rubber belts. Boiled linseed oil is good; also equal parts of black lead, red lead, French yellow, litharge and enough japan dryer to make it dry quickly. This will give a smooth polished surface.

MISCELLANEOUS NOTES

Lead Work in Metallurgical Construction (By H. T. Durant).—In many metallurgical works, notably those connected with the wet metallurgy of copper and zinc, there is a considerable amount of lead work, comprising lead pipes and lead-lined launders and tanks. The lead used, whether in sheet or pipe, should be that which is usually known in the trade as "chemical lead," which is free from copper and zinc or other elements, which would in the presence of an electrolyte set up a couple and thereby cause failure and leaks at certain spots.

For a similar reason burnt joints are far preferable to wiped ones, for in the case of the latter the two metals, lead and tin, are in contact with the acid solutions and further than this, it will be found that wiped joints are often porous and will slightly leak or sweat. Another objection to wiped joints is the doubtful percentage of tin in some bought solders, though it is usual to make one's own solder where much wiping has to be done.

Lead burning should, for economic motives, be done when possible with compressed oxygen, instead of with atmospheric air; should coal gas also be available, then the necessity of wastefully generating hydrogen from zinc and acid is avoided. Generally speaking, plumbers who are accustomed to domestic or house work cannot be employed with economy in this class of work; it is more satisfactory to get lead burners, who have been trained at sulphuric-acid or similar works.

Lead pipes, owing to their irregularity, are liable to air locks, which must be guarded against. All lead pipes should, when in position, be carried on timber runners suitably hung or supported, and so that the weight of the pipe is borne evenly along the whole length—the pipe is partly encircled about every 6 ft. by a strap of lead spiked at each end to the timber; this holds it in position on its carrier. In the case of pipes up to 2-in. bore, set vertically or approaching the vertical, these lead straps should be burnt to the pipe and should be strong enough to prevent the downward creeping of the lead. Where larger pipes have to go up vertically, it is preferable to put the pipe in in flanged lengths, as the flanges form the most convenient means of taking the weight to prevent creeping or collapse of the pipe. A pipe when hanging vertically and unsupported at its lower end, as in the case of a pump suction, elongates considerably unless the weight is properly taken by one of the above-mentioned

methods. Bends, elbows, etc., of above 2-in. bore are always best made flanged at each end. They can then be easily held to shape by a suitably bent angle iron, so that when coupling up, each end of the angle iron will engage with one of the flange bolts, and thus the angle iron will serve as a stffener.

Lead pipes should not be buried unless unavoidable, and if buried they should be boxed or otherwise protected from any weight or pressure. The usual weight of pipes suitable for ordinary works purposes and capable of withstanding an internal pressure up to about 25 lb. per sq. in. is 10 lb. per yd. for every inch internal diameter.

Pipes, where permissible, naturally cost less than lead-lined launders, but the transport of lead pipes to any considerable distance is more costly than sheet lead, on account of the packing required. All pipes should be protected when placed in positions where they can be trodden on or otherwise damaged.

Subsequent to the construction period it will be found a great advantage if pipes have been laid flanged in convenient lengths. In general the flanged length of lead pipe should be of such a weight that it can be easily handled on its wooden runner by two men. Small piping up to 2-in. bore is supplied in coils, containing about 60 ft., which are easily straightened out and rapidly laid; if this be done and the pipe is not laid in flanged lengths, the result is a saving of expense in construction, but an increased operating expense and trouble when the pipe gets blocked or for any reason requires opening.

Flanges are easily cast from lead containing 8 to 10% of antimony; these conform to standard dimensions, but the minimum thickness of metal in the cast flange should be not much less than ½ in.; they are drilled as usual for coupling, and are sufficiently hard to be bolted up with bolts and washers without using any stiffening plate. The flanges are burnt to the end of the lead pipe, the flange boss being cast slightly tapered inside so that the pipe seats well into it, and the burning is done so that the pipe is burnt to the flange at the outside of the boss, and again inside where the pipe goes almost through the flange. Iron bolts can always be used for coupling up as they do not come in contact with acid or harmful solution; the flanges as cast only require a rough facing with a milling file. If a pipe in a system where joints are not flanged needs to be opened up, it must be sawed through, probably in the wrong place to start with, then joined again by burning or wiping.

The sheet lead required weighs, according to its service, from 7 to 10 lb. per sq. ft. Most lead rolling mills can turn out any width up to 7 ft. 9 in. It is an economy, especially in the case of launders, to get the lead the right width, as it saves time, labor, cutting and waste. It is obvious, though a precaution sometimes neglected, that all loose or projecting

nails and wood splinters should be removed before the sheet lead is laid in position; similarly, any bolt heads should be countersunk and all dirt or anything which can wound the lead should be first removed; also workmen on the job should wear slippers, or at least not boots.

Water Softening.—In many of the mining districts in Western Australia the only available water for boiler feed contains a large amount of scale-forming constituents. Gordon F. Dickson, in the *Monthly Journal* of the Chamber of Mines of Western Australia describes a process of water softening, and says that with a comparatively small expenditure a plant can be erected that will give excellent results.

The plan, Fig. 185, shows the arrangement of a plant in use at the

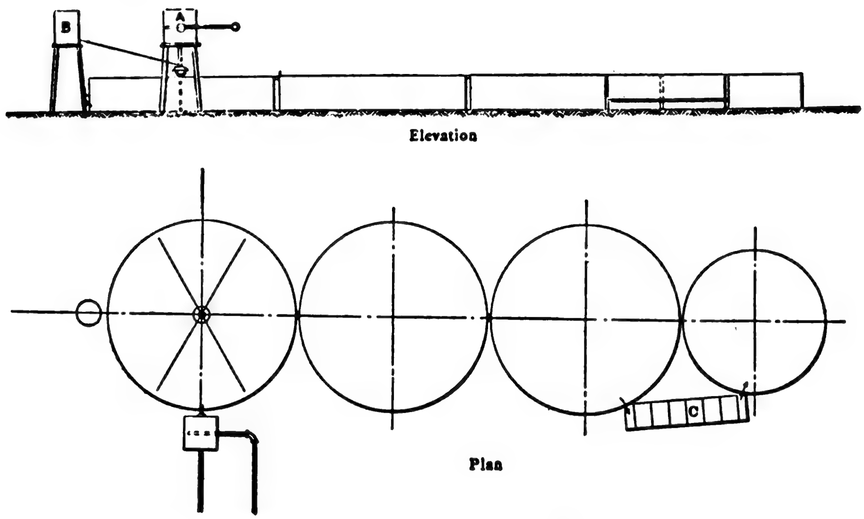


FIG. 185.—GENERAL ARRANGEMENT OF WATER-SOFTENING PLANT.

Northern mines, Lawlers. The chemical receiver *B* is capable of holding a supply of dilute chemical reagent, the flow from which is regulated to last 24 hr. The first or separating tank is fitted with a central column in which is a spindle and bowl of the Butters automatic distributor type, and into this the hard water, after passing through the heater *A* is delivered, where it mixes with the required quantity of solution fed from the chemical receiver *B*. The distributor, as shown in Fig. 186, has six arms, each perforated with a number of holes at equal distances and placed at an angle of 45 deg. below the center line of the pipe, through which the combined water and chemical solution is sprayed into the tank. The rapid circulating motion of the arms causes a certain agitation which allows of complete mixing, and quicky effects a separation of the solids.

To provide for thoroughly cleansing the pipes of any adhering scale

the ends of the distributor arms are fitted with ordinary screw plugs. By removing these daily, and passing a light iron rod through the pipes, the latter are kept perfectly clean. To prevent the water from getting on the bearing surfaces of the spindle and bowl, a hood is fitted around the top of the bowl, which carries off any water that might overflow and be liable to cause incrustation on the working parts. Besides the increase in temperature gained from passing the hard water through the heater *A*, exhaust steam is also led into the separating tank, and the temperature is further raised to 160° F. This removes a percentage of the carbonates of lime and magnesia contained in the water, and further effects a complete blending of the chemical reagent with the hard water.

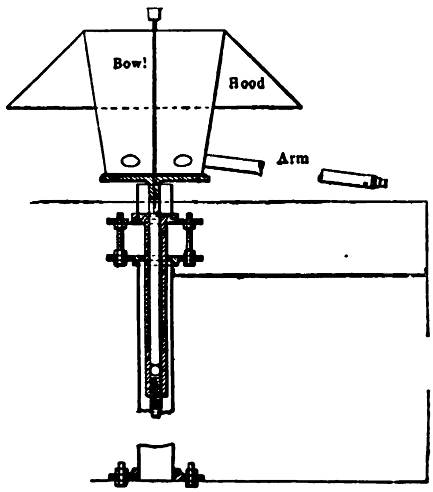


FIG. 186.—DETAILS OF DISTRIBUTION, WATER-SOFTENING PLANT.

The water overflows from the first to the second tank, where the principal settlement takes place, thence to the third tank for further settlement, after which it passes through the filter *C*, where any of the precipitate that has passed the settling tanks is caught and any oil introduced with the exhaust steam is removed. The water before treatment contains 16° total hardness, 5 of which are permanent and 11 temporary. After treatment, daily tests show the total hardness of the water to average less than 2°. Analysis of the water shows the following constituents in parts per 100,000: Organic and volatile (including CO_2), 6.07; silica, 7.81; ferric oxide and alumina, 3.63; calcium sulphate, 8.30; calcium carbonate, 4.57; magnesium carbonate, 9.40; sulphuric oxide, 1 80; chlorine, 7.32; total, 48.90.

The softening of the water is effected by the addition of 10 lb. of caustic soda per 24 hr. for 19,000 gal. of feed water. Samples of water taken

from the flow entering the boiler-feed tank show, on analysis, only traces of lime and magnesia. The plant has been in operation for four months and, except for occasional stoppages to remove the accumulated precipitate from the tanks and the brief time required each day for cleaning the distributor arms, it has worked continuously and has given the utmost satisfaction, at a trivial cost for labor and maintenance.

Air Lift for Transporting Sand.—An application of the Pohle air lift to raising and transporting the sand from cyanide tanks was recently made at Burbanks Main Lode mine in Western Australia. This interesting operation is described in the *Monthly Journal* of the Chamber of Mines of Western Australia. The sand had to be carried across a public road, requiring a lift of 23 ft. 3 in. It was decided that a submergence of 63% of the total length of water pipe was required, which is a single-stage lift would have required a pit 39 ft. 6 in. deep. For this reason a two-stage lift was adopted, reducing the depth of pit required to 15 ft. 6 inches.

The air supply for each lift is regulated by floats, so that a nearly constant level is maintained in each head box, and no air is blown to waste. The two lifts combined use 47 cu. ft. per min. of air; discharge 320 lb. sand per min. and require 800 lb. water per min. This means 6.8 lb. sand are discharged per cu. ft. of air. The air used was drawn from a large receiver. In testing the air lift, this receiver was filled to a gage pressure of 70 lb. per sq. in. and was then shut off from the compressor, and the fall in pressure in a given time noted. During this time no air was drawn from the receiver for any other purpose. Mine water was used for sluicing, and the amount available was only little more than sufficient to carry away (along a V-shaped launder having a fall of $\frac{7}{8}$ in. per ft.) as much sand as one man could shovel into the discharge door of the vat. The sand is heavy, and rapid motion is necessary throughout to prevent it settling; the proportion of water to sand is about $2\frac{1}{2}$ to 1.

Before the air lift was put in, it was the practice to shovel the sand through the bottom discharge doors into trucks; with the air lift one man empties the vat in considerably less time than two did with the trucks, no time having to be lost in changing and waiting for trucks. The vats hold 30 long tons and, with one man shoveling, they are emptied in $3\frac{1}{2}$ hours, using 45 to 47 cu. ft. of free air per min. Each discharge door has a 4-in. socket screwed into it, and the door is never removed. From the bottom of the socket a pipe leads down vertically, with an elbow just above the launder; inside the vat a pipe is screwed into the top of the socket long enough to reach above the top edge of the vat. To discharge the vat the pipe projecting through the sand is unscrewed and pulled out, leaving a convenient hole through which to start sluicing. To avoid the risk of having more sand sluiced into the launder than the

water is able to carry along, from 25 to 30% of the total added water is supplied to the highest end of the launder running under the discharge doors.

Both sections of the lift are of the same dimensions, except in height and consist of a 6-in. cast-iron pipe (which constitutes the well) with a 3-in. wrought-iron pipe inside, extending from within 4 in. of the closed bottom end of the 6-in pipe, up to the required height of lift. The air pipes are ¾ in., though ½ in. would be ample; they are carried down the pit outside the 6-in. pipes, and are turned up at the bottom through the center of the dead end at the bottom of the 6-in pipes. They extend upward for 10 in. in the 6-in. and for 6 in. in the 3-in. pipes. The nozzle ends of the air pipes are closed and twelve hacksaw slits are cut across each of the ends, the cuts extending round about one-quarter of the circumference of the ¾-in. pipe and cut at an angle of 45 deg. thus giving the air an upward direction.

The lift has worked well from the start and has caused no trouble or delay. When the water supply fails, a ½-in. water pipe (which is kept connected by flexible hose to a supply having a pressure of about 50 lb. per sq. in.) is passed down the 6-in. pipe which is cleared almost as fast as the ½-in. pipe can be lowered. Two or three minutes are sufficient to clear it at any time. The only wearing parts are the 3-in. pipes, and these, having no bends, are likely to last a long time. The consumption of air may be taken as about 50% more than the best pump would use when in perfect condition. Unlike sand pumps, however, the air lift remains in perfect condition. The added water is heated to a temperature of 95° F. by passing a part of it over exhaust-steam pipes on its way to the lift. This heat, no doubt, adds to the efficiency of the lift.

Measuring Small Quantities of Low-pressure Air.—It is frequently desirable to measure the quantity of air used in metallurgical operations, as in air agitation. In order to accomplish this easily, and without resort to expensive apparatus, G. S. Weymouth, of the Great Fingall mine, has designed the apparatus shown in Fig. 187 as described in the *Monthly Journal* of the Chamber of Mines of Western Australia.

The method consists in substituting temporarily an equal flow of air under conditions which allow of easy measurement. The only disadvantage is that the flow of air must be diverted from the point of application for half a minute, while the measurement is being made. In the figure *A* is a valve regulating the air to the required pressure; *B* is a mercury gage; *C* and *D* are stop-cocks; *E* the apparatus under test; *F* a small tin receiver connected by rubber hose to *D*; *G* is a round orifice in a tin plate for air outlet, and *H* is a water gage with a range of about 12 in. to allow for occasional rushes of air while manipulating the cock *D*.

A low-pressure supply being required at *E*, say 5 lb. per sq. ft. and the

cock C being open and D closed, the regulating valve A is opened until the gage B indicates the required pressure. The valve A should not be touched again during the test. The height of mercury on B is marked; and to measure the quantity of air flowing at this instant, the cock D is opened and C closed; D is then gradually closed until the gage B reads the same as before. In this way the total substituted resistance to the free flow of air is equal to that of the apparatus being tested, the pipe A to I in both cases acting as an intermediate receiver with constant in- and out-flow as provided by the gages. The height of the water gage is now taken, and the area of the outlet orifice noted; the cock C is opened and D closed, the gage B reading the same as before. This is a check on the constancy of flow during the testing period. The velocity through

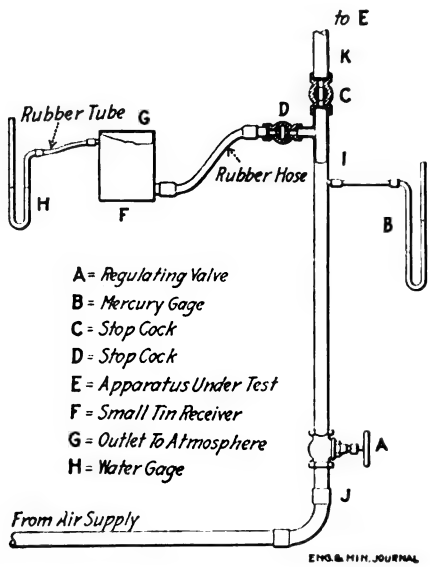


FIG. 187.—DEVICE FOR MEASURING LOW-PRESSURE AIR.

the orifice G is that due to a head of air equivalent to the pressure on the water gage, and is $\sqrt{2gh}$ = ft. per sec., h being the height of the motive column (head) in feet. Reducing to inches of water, and simplifying, this becomes $\sqrt{\text{in. water}} \times 66.1$ = ft. per sec. The coefficient of flow for such an orifice being 0.64 owing to the contraction of the flow near the edges, the actual mean velocity over the full size of the orifice becomes $\sqrt{\text{in. water}} \times 42.3$ = ft. per sec. Pipe sizes do not come into the calculations, provided they are large enough to carry the quantity of air. The receiver F need be only of sufficient size to reduce the kinetic action of the air to a negligible quantity and give a true reading on the water gage.

In the author's actual experiments the receiver F was a tin of about

1-qt. capacity. For air-lifts consuming about 50 cu. ft. per min. a 5-gal. can was used with orifices of 0.5-, 1-, 2- and 4- sq. in. area. The pipe fittings in this case were 1 in. No difficulty was experienced in measuring quantities between 0.5 and 60 cu. ft. per min. By substituting a high-pressure gage at *B* readings can be taken at any pressure. A little experience soon gives the proper size of orifice to use.

VI

HYDROMETALLURGICAL PROCESSES

PRELIMINARY TESTING

Graphic Method of Illustrating Extraction Tests (By H. K. Picard).—There is herein suggested (*Bull.* 89. I. M. M.) a graphic method of reporting the results of extraction or concentration tests on ore samples, which has been found particularly suitable when a large number of products result. The method has been in use in my laboratory for some time past, and is generally approved by those interested. So far as I am aware, nothing similar has been published, though it is quite possible that like devices are employed in other laboratories and works. .

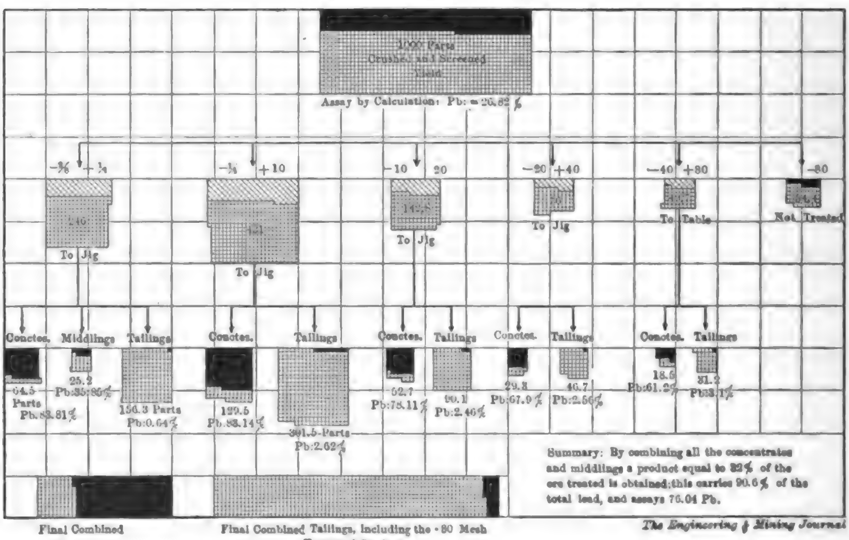


FIG. 188.—CHART SHOWING RECOVERY OF LEAD CONCENTRATES.

As an example I give a concentration test on a lead ore where the number of products is not excessive; even in this simple case it will be admitted that the graphic method is easier to follow than would be a list of figures detailing the results of such a test. For instance, a middling product which assays 30% lead at once gives the impression that it is too rich to be ignored and will require re-treatment. But reference to the graphic statement shows that the amount of this product is so small as to render

re-treatment inadvisable or unnecessary, and that it may be mixed with the bulk of the concentrates without appreciably lowering their value.

A sheet of squared paper divided into tenths is most suitable for the diagrams required in the graphic method of illustration. Each square is taken to represent a convenient weight of ore, say 1 lb. or 1 gram, according to requirements. A parallelogram is drawn containing a number of squares equal to the weight-units taken for the test. In the example given 1000 lb. of ore were taken, and are shown in Fig. 188, by 1000 squares at the top of the sheet. The tenor of the ore in lead is represented within this figure by a wash of solid color occupying squares equal to the amount of metal present. In this instance the ore assays 26.8% lead, and therefore 268 squares are colored. The sample was then crushed and graded into the products shown, the size of each figure again being proportional to the weight. The lead tenors of each grade are similarly shown, except that, not being final products, the amounts of metal are represented by cross-shading. The −80 grade was not further treated, and thus became a final product; its lead contents are therefore shown in solid color.

Following the diagram, each grade was separately concentrated, yielding the products indicated—these again being shown relatively to their weights; and, being final products, their lead contents are represented in wash. All the lead originally present is thus divided up and shown in the final products, the sum of whose weights and contents is equal to that in the original ore taken.

In the example, purposely selected for its simplicity, there is obviously no difficulty in deciding what to do with the products; but with complex ore yielding various middlings of differing values, the diagrammatic method will be found of assistance in deciding whether such products should be rejected, re-treated, or mixed with some other product. Two or more metals can be shown by employing different colors. With gold or silver ores the amount of metal would be too small to show proportionately, but the relative amounts may be indicated if some convenient unit be selected. Thus the small squares may be taken to represent 1 grain, 1 dwt. or 1 gram per ton, according to the richness of the ore or concentrate.

The size of the unit to be adopted in any particular case will be determined by the smallest product to be shown. This must be of appreciable dimensions, and should conveniently occupy not less than ten squares. A unit square of $\frac{1}{10}$ in. has usually been found suitable. The suggested method is applicable to operations other than concentration, though it is in regard to the latter that it has been found specially useful.

Study of Leaching Processes.—A basis for the study of leaching processes is explained in the September, 1913, issue of the *Colorado School of Mines Magazine*, by Robert B. Elder, of the Chiksan Mining Co., Chosen.

According to this system, the solution from a leaching vat is measured for successive short periods of time, samples being taken meanwhile, which are assayed and analyzed. The total leaching period of the charge is covered by the samplings, each of which, however, should be of short duration in order to differentiate the constantly changing rates of solution.

Results from a set of samples covering the treatment of a charge may be plotted, as shown in the accompanying diagram, Fig. 189, different colors being used to represent the different elements. Plotted thus, the

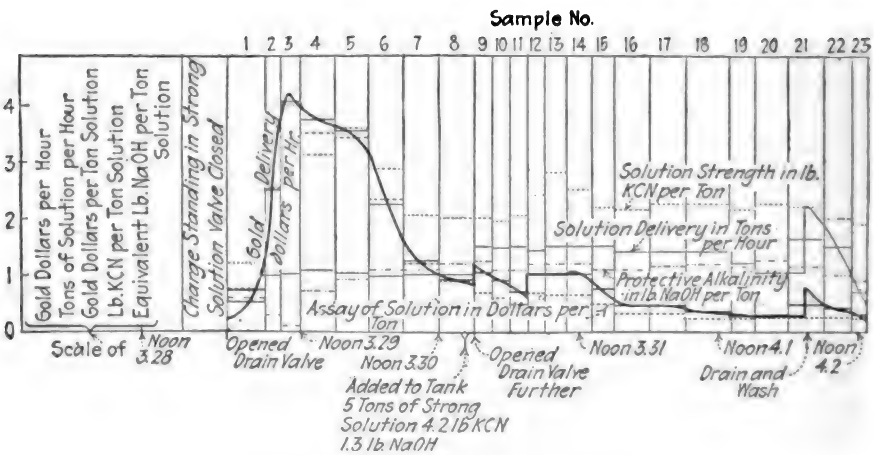


FIG. 189—DIAGRAM OF TEST RESULTS.

ordinate of any point on the gold-production curve represents the rate in dollars per hour at which the gold comes from the tank, and the area under the line represents the amount of gold delivered. This is, of course, upon the supposition that the rate of change in assay is a constant in the period between taking consecutive samples. However, it is plain that while this condition is not exactly fulfilled, the error is negligible.

Suppose the cost of allowing the percolation to continue be ascertained and plotted on the sheet in terms of the same units as the gold production. The time at which the percolation should be discontinued and the tank recharged in order to obtain the greatest net profit per ton under those conditions of operating, is at the point of intersection of the gold-extraction line and the cost line. In order that this may be strictly true, however, the gold-extraction line must be corrected to allow for the losses and cost of precipitation and melting.

At the cyanide plant of the Chiksan Mining Co., Chosen, treating accumulated tailings from a 40-stamp mill, the information derived in this way has enabled us to increase the extraction from 75 to 85% on $2 material at an increased treatment cost of 5c., giving, therefore, an addi-

tional net profit of 15c. per ton treated. The present cyaniding cost is \$0.44 to \$0.46 per ton.

The ore here is quartz carrying free gold and small amounts of auriferous pyrite. It is stamped through 40-mesh screen, amalgamated, and concentrated on Standard tables. The tailing is held in two settling ponds and the mill water, containing slime equal to about 12% of the original ore, is run to waste. The tailing from these settling ponds is loaded into the cyanide leaching vats by native coolies. Lime equivalent to 1½ lb. CaO per ton of ore is added with the charge.

Explanation of the diagram: At 6 p. m., Mar. 18, the changing was completed and solution carrying 3.5 lb. KCN and protective alkalinity, equal to 1.3 lb. NaOH per ton, was added till the charge was covered. This was allowed to stand till midnight, when the drain valve was opened. As the solution was drawn off, solution carrying 2 lb. KCN and 1.2 lb. protective alkalinity was added, so that the charge was always covered. Between the hours of 4 and 5 p. m. on Mar. 30, instead of weak solution 5 tons of strong solution were added. This began to appear in the discharge about 10 hr. later and its results are apparent in the lifting of the KCN, gold delivery and assay-value lines.

Preliminary Testing Work at Yuanmi Mill, Western Australia.—Before designing the new mill at the Yuanmi mine, 60 miles south of Sandstone, Western Australia, the Yuanmi Gold Mines, Ltd., carried on comprehensive experimental work at its other mill at the Oroya Black Range mine. A large quantity of representative ore was carted to the latter mill, where a five-stamp battery and amalgamating table were set apart for the purpose. Some of the typical results of this work, as reported in the *Monthly Journal* of the Chamber of Mines of Western Australia, are given below.

The oxidized ore in sight is a soft, yellow, kaolinized, easily crushed lode matter, mixed with a hard siliceous matrix, difficult to grind. When the ore was crushed to pass a 30-mesh screen in a battery of five, 1250-lb. stamps, running at 102 drops of 6½ in. per min. and with a 3½-in. discharge, the stamps had a duty of over 8 tons per 24 hr. The pulp from the stamps passed over ordinary stationary amalgamated copper plates, an extraction of 48.1% being obtained. The tailing after amalgamation was separated by decantation into sand and slime yielding 35% of sand and 65% of slime. The sand assayed 55s. per ton and the slime 21s. per ton.

After leaching the sand with an alkaline cyanide solution the tailing assayed nine shillings per ton. This residue was reground to pass a 150-mesh screen and agitated with cyanide solution which brought the tailing down to 3s. per ton. The slime, after a short agitation, yielded a residue of 2s per ton. The consumption of chemicals was in all cases low

and aëration was not found to increase the extraction. These results indicated that a 90% recovery could be expected in practice from the oxidized ore of this grade. It was evident also that leaching of the sand would not prove an economic success and that an all-sliming plant was likely to yield the most profit.

In order to insure an adequate circulation of cyanide solution, careful slime-settlement tests were conducted. These tests were carried on in a medium of similar character as regards temperature, density and contained chemicals to that likely to be encountered in practice. To this end a system of curves was plotted, the ordinates being the percentages of solid material and the abscissas the time required for settling, showing the settling ratio of the slime on various mines. These settlement curves were made during work under identical conditions as to height and diameter of column, primary percentage of solids in pulp, etc., and were made in the actual solution in use in the various plants to which they refer. By comparing the Yuanmi settlement curve with curves obtained from mines with existing efficient settlement area, it became possible to estimate the plant required at Yuanmi for the settlement of a given tonnage. This method of obtaining exact and reliable comparative settlement data was of great value and a settlement plant was designed from this information. The results fully justified the method.

CYANIDATION OF ORES

Poisoning by Cyanide.—The committee of the Mining Regulations Commission of Transvaal, appointed to look into the matter of poisoning by cyanide, circularized mine managers, cyanide managers and the leading metallurgists in the Transvaal. From 55 replies received, the following conclusions were reached: (1) That cyaniding is *per se* a healthful occupation; (2) that the number of fatal cases of cyanide poisoning by drinking is comparatively small and attributable almost without exception to carelessness or pure accidents; (3) that cyanide eczema, occasionally noticed among those who have to handle zinc shavings in the precipitation boxes, is unknown where the cyanide solution is replaced by water previous to the zinc shavings being handled; (4) that many cases of "gassing" occur among those who are engaged in treating the gold slimes from zinc-extractor boxes by the acid process, on account of the hydrocyanic acid gas and the arseniuretted hydrogen given off.

The necessary preventive measures suggested by the foregoing considerations include the following: (1) The provision of an adequate supply of wholesome drinking water about plants and assay offices, the same to be distinctly labeled "Drinking Water"; (2) the replacement of strong cyanide solution used in the precipitation boxes by water previous

to the zinc being handled; (3) the effective hooding of the dissolving bath in which the gold precipitates are treated, and the use of some form of mechanical agitation in order to eliminate the necessity of raising the hood.

It is recommended, that boxes labeled "Antidotes for Cyanide," with directions for use affixed to the lids of the boxes, should be kept in prominent and easily accessible parts of the cyanide plants. Each box should contain: A spoon and a metal receptacle to hold about 1 pt.; one blue hermetically sealed vial containing 30 c.c. of 33% solution of ferrous sulphate; a white vial containing 30 c.c. of caustic potash, and one package of oxide of magnesium (light). The directions for the use of the antidote should be as follows:

Preparation of Antidote.—Quickly empty the contents of the blue vial, of the white vial, and of the magnesia package into the metal receptacle, and stir well with the spoon. This should be done as rapidly as possible, as the patient's chance of life depends on promptness.

Administration of the Antidote.—If the patient is conscious make him swallow the mixture at once, and lie down for a few minutes. If the patient is not conscious, place him on his back and pour the mixture down his throat in small quantities, if necessary pinching his nose in order to make him swallow.

Incite Vomiting.—After the antidote has been given, try to make the patient vomit by tickling the back of the throat with a feather or with the fingers, or giving a tumblerful of warm water and mustard.

Sodium vs. Potassium Cyanides.—The Roessler & Hasslacher Chemical Co. is calling attention to the various grades of cyanide it now puts on the market, and makes some calculation on their relative efficiencies. The results will doubtless be of interest to cyanide operators and are shown in Table XXXIV. The sodium cyanide is packed in 100- and 200-lb. cases. There is, of course, a freight saving on the higher grades also.

Table XXXIV.—Relative Efficiencies of Commercial Grades of Cyanide

Compound	Commercial name	CN %	Price, c. per lb.	Cost of CN, c. per lb.
Sodium cyanide............	129%	52	...	51.25
Sodium cyanide............	100%	40	20½	51.25
Mixed cyanides............	98–99	39.5	21	53.16
Potass. cyanide............	95–96	38.4	24	62.50

Making up Solutions (By H. T. Durant).—In making up cyanide and other solutions to any required strength, various tables are sometimes used. For ordinary mill and works purposes, the necessary calculations can be done mentally by the shiftmen. One ton is 2000 lb., and, there-

fore, 1 lb. of anything to 1 ton represents 0.05% (one-twentieth of 1%). For example, a tank contains 300 tons of solution, testing 0.04% cyanide, and it is required to raise the strength by the addition of solid cyanide to, say, 0.10%. Remembering that 1 lb. added to 1 ton gives 0.05%, then it is obvious, the required increment being 0.06%, that 1⅕ lb. of solid cyanide must be added to each ton, and that, therefore, 360 lb. (300 × 1⅕ = 360) of solid cyanide will be required to raise the above-mentioned 300 tons from 0.04% to 0.10%.

An alteration of the above rule would be required in the event of the ton not consisting of 2000 lb., or in event of high-grade sodium cyanide being used while still adhering to the older method of reporting it in terms of potassium cyanide, or again if the cyanide were merely reported in terms of cyanogen regardless of the base with which it is combined.

Rapid Estimation of Pulp in Cyanide Tanks (By Mark R. Lamb).—Table XXXV presented herewith is compiled as an aid to the calculation of the pulp contents of cyanide tanks and should be extremely useful to all who are in actual contact with cyanide-plant operations. All calculations in the table are based on the weight (in grams) of the liter of pulp, placing the specific gravity of the slime at 2.5.

The tonnages per foot of depth cover various sizes of slime tanks and may be made to include the intermediate sizes if thought desirable, but as the internal diameters of wooden tanks are not ordinarily made to even feet and as steel tanks are not always made exactly to specified dimensions, each operator must calculate a column to fit his own tanks. Such a column can be interpolated easily and quickly by using a slide rule, setting the area of any tank over the area of one of the tanks in the table and reading the desired capacities per foot depth over the various capacities given in the table for various consistencies. A space is left for writing in an additional column.

The figures could have been worked out to a greater accuracy but as the measure of the depth of slime in the tank is rarely made closer than within an inch and as the sample weighed usually contains less slime than the real average of the charge owing to the settlement in the tank and in the sampling bucket or dipper, the accuracy of the table is much greater than that of observation. The figures opposite the lower percentage of the moisture are, of course, only accurate with saturated pulps. I am indebted to a similar tabulation by E. M. Hamilton for the idea and for some of the figures.

Calculator for the Cyanide Plant.—Several articles have from time to time been published in the *Eng. and Min. Journ.* describing report froms and similar devices to aid in the control of operations at cyanide plants, but no mention has been made of the adjustable calculator shown in Table XXXVI. This table has been used at the Homestake plant in

TABLE XXXV.—PULP TABLE FOR CYANIDE TANKS

Tons of dry slime per foot of depth of tank — Diameter of tank, feet: 12′	16′	20′	24′	28′	30′		Wt. of liter in kilos, sp. gr.	Weight per cu. ft., tons	Cu. ft. of pulp per ton	Cu. ft. of pulp per dry ton of slime	Water in pulp, %
0.07	0.125	0.20	0.28	0.38	0.441	....	1.012	0.0316	31.6	1600	98
0.145	0.26	0.40	0.58	0.78	0.907	....	1.024	0.032	31.2	780	96
0.22	0.39	0.61	0.88	1.17	1.37	....	1.037	0.0324	30.8	515	94
0.30	0.53	0.83	1.18	1.6	1.86	....	1.050	0.0328	30.4	380	92
0.38	0.67	1.05	1.5	2.02	2.36	....	1.064	0.0332	30	300	90
0.46	0.81	1.27	1.82	2.45	2.85	....	1.077	0.0337	29.7	248	88
0.54	0.956	1.5	2.15	2.9	3.36	....	1.092	0.0341	29.3	210	86
0.63	1.12	1.75	2.5	3.36	3.92	....	1.106	0.0346	29.9	180	84
0.71	1.25	1.95	2.82	3.80	4.42	....	1.121	0.035	28.5	160	82
0.80	1.44	2.24	3.22	4.33	5.05	....	1.136	0.0355	28.2	140	80
0.90	1.59	2.5	3.57	4.8	5.6	....	1.152	0.036	27.8	126	78
0.99	1.76	2.76	3.96	5.32	6.20	....	1.168	0.0365	27.4	114	76
1.09	1.93	3.02	4.34	5.83	6.80	....	1.184	0.037	27	104	74
1.19	2.12	3.3	4.76	6.4	7.45	....	1.201	0.0375	26.6	95	72
1.29	2.29	3.6	5.15	6.9	8.04	....	1.219	0.038	26.2	88	70
1.41	2.50	3.9	5.64	7.6	8.84	....	1.237	0.0386	25.8	80	68
1.51	2.68	4.2	6.02	8.1	9.44	....	1.255	0.0392	25.4	75	66
1.62	2.90	4.5	6.45	8.7	10.10	....	1.275	0.0398	25	70	64
1.74	3.08	4.85	6.95	9.3	10.85	....	1.295	0.0404	24.7	65	62
1.86	3.30	5.15	7.42	9.9	11.6	....	1.315	0.0411	24.3	61	60
1.98	3.53	5.5	7.93	10.6	12.4	....	1.337	0.0417	23.9	57	58
2.10	3.73	5.8	8.37	11.2	13.1	....	1.358	0.0424	23.5	54	56
2.26	4.02	6.18	9.05	12.12	14.15	....	1.381	0.0431	23.1	50	54
2.40	4.28	6.7	9.6	12.9	15.02	....	1.404	0.0438	22.8	47	52
2.51	4.47	7	10	13.45	15.70	....	1.429	0.0445	22.4	45	50
2.7	4.78	7.5	10.7	14.4	16.8	....	1.453	0.0454	22	42	48
2.86	5.03	7.85	11.3	15.3	17.7	....	1.479	0.0462	21.6	40	46
2.98	5.30	8.3	11.9	16	18.6	....	1.506	0.0470	21.2	38	44
3.14	5.57	8.75	12.5	16.8	19.6	....	1.533	0.0479	20.8	36	42
3.33	5.92	9.25	13.3	17.8	20.8	....	1.562	0.0488	20.4	34	40
3.54	6.28	9.85	14.1	18.9	22.1	....	1.592	0.0497	20	32	38
3.65	6.43	10.4	14.6	19.5	22.8	....	1.623	0.0507	19.7	31	36
3.91	6.94	10.9	15.6	20.4	24.4	....	1.655	0.0517	19.3	29	34
4.03	7.17	11.3	16.1	21.6	25.2	....	1.689	0.0528	18.9	28	32
4.20	7.45	11.7	16.7	22.3	26.2	....	1.724	0.0539	18.5	27	30
4.54	8.05	12.6	18.1	24.2	28.3	....	1.760	0.055	18.2	25	28
4.70	8.36	13.1	18.8	25.2	29.4	....	1.798	0.0562	17.8	24	26
4.92	8.73	13.7	19.5	26.3	30.7	....	1.838	0.0574	17.4	23	24
5.51	9.13	14.3	20.5	27.5	32.1	....	1.879	0.0587	17	22	22
5.38	9.56	15	21.5	28.8	33.6	....	1.923	0.0601	16.6	21	20

Note.—Column left blank for insertion of exact figures representing tank in use.

TABLE XXXVI.—CALCULATING TABLE FOR BRINGING CYANIDE SOLUTIONS UP TO STRENGTH

Depth of effluent solution in sump in feet[1]	Desired working strength[2]	Strength of effluent solution in sump to which cyanide is to be added until the desired working strength is obtained								
(Cut with knife along dotted lines; then insert "sliding strip" shown below the table.)										
		Pounds of cyanide to be added to solution in sump								
1		3	6	9	12	15	18	21	24	27
2		6	12	18	24	30	36	42	48	54
3		9	18	27	36	45	54	63	72	81
4		12	24	36	48	60	72	84	96	108
5		15	30	45	60	75	90	105	120	135

[1] Figures in this table are calculated for a sump holding 30 tons per foot depth.
[2] By strength is meant the percentage of free KCN in the solutions.

Sliding strip

Percentage of KCN in solutions										
0.120	0.115	0.110	0.105	0.100	0.095	0.090	0.085	0.080	0.075	0.075

South Dakota, and has been found to be of great convenience when conducting experiments involving variations in the strength of cyanide carried by the ongoing solutions.

The illustration is almost self explanatory. A simple table having been calculated, a single vertical column, equal in width to those following it, is ruled at the left of the columns containing the indicated weights of cyanide. The headings of the latter columns are entered on a detached strip of paper which is inserted in guides cut in the main sheet. By shifting this strip to left or right the headings of the various columns are shifted while the positions of the headings, relative to one another, are not changed. Provided that the columns are prepared for uniform variation of effluent strength, the column at the left may be headed "Working Strength Desired" and the table becomes adjustable for any working strength within the range selected.

Heating Cyanide Solutions (By John Tyssowski).—In the Montana. Tonopah mill, Tonopah, Nev., heating the cyanide solutions during agitation is practised. Hendryx agitators are used and at first radiators were put in these. After a couple of months a scale formed over the steam pipes rendering their efficiency extremely low. The practice finally adopted is to pass steam at about 30 lb. pressure into the solution through a 1-in. pipe terminating below the surface of the pulp. Solutions are now kept at 90 to 95° F. (higher temperatures cause an increased cyanide consumption) and with a small consumption of steam, and no additional cyanide, the extraction has been bettered almost 5 per cent.

Aëration of Cyanide Solutions (By John Tyssowski).—It is a recognized fact that the dissolving of gold or silver in cyanide can only be effected in the presence of excess oxygen. It is, therefore, of prime importance to provide this oxygen while solution is being effected. When a solution becomes spent, *i.e.*, excess oxygen has been used up and excess of hydrogen results, the solution becomes positively charged, whereas it should be negative in order that the metal to be dissolved, which has a positive charge, shall go into solution. It is the function of aëration to correct this condition. It also serves to break up and eliminate objectionable cyanicides, ferrocyanides, arsenious acids, etc.

In the Nova Scotia mill at Cobalt, Ont., especial attention is paid to aëration of the cyanide solution, and this is particularly necessary on account of the use of the Trent agitator, in which the pulp is kept in circulation by pumping solution drawn from the top of the tank through the radiating arms of a distributor set at the bottom of the tank. The arms of the agitator are bent, as in a rotary lawn sprinkler, so that the reaction of the water being forced through them causes rotation. Where Pachuca agitating tanks are used, the air for agitation furnishes the necessary oxygen.

In Fig. 190 is shown the arrangement used in the Nova Scotia mill to aërate the solution from the Moore filters as it passes to a clarifying tank. The end of the discharge pipe is turned up and terminated in a nozzle having small perforations. The solution is sprayed through this and falls into a box with holes bored through its bottom, which floats

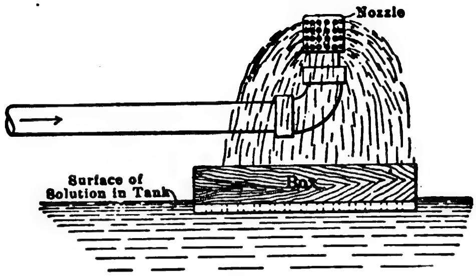


FIG. 190.—SPRAY FOR AËRATING SOLUTION IN CLARIFYING TANK.

on the surface of the solution in the tank. This arrangement not only serves to aërate the solution, but aids materially in clearing, no filter press being used to clear solutions further before zinc dust for precipitation is added.

The silver solution from the clarifying tank is joined by the overflow from three Dorr settlers and aërated by being run over the arrange-

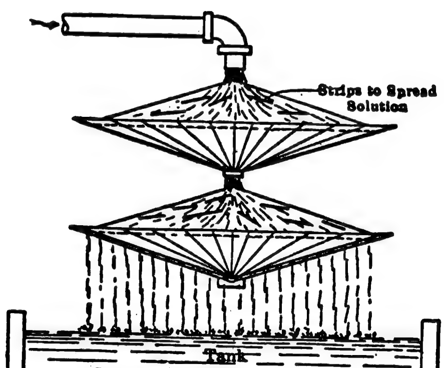


FIG. 191.—AËRATING ARRANGEMENT USED IN O'BRIEN MILL.

ment shown in Fig. 191 below which in the tank is liberated fresh air delivered from cowls set on the roof of the mill building. The solution is run on a convex-shaped surface covered with strips so as to spread it. From this it drains on a concave surface placed below, but with its edge extending beyond that of the upper one. The solution drains to the center of this surface, where it is discharged to another convex surface below, and so on until it has passed over about six. The lowest one is

perforated so that the solution trickles to the tank below. A sheet of canvas extending from the top of the tank to above the aërating arrangement is stretched about the latter so as to confine the air delivered from the cowls and force it to pass up between the spreading surfaces, thus affording ample opportunity for the absorption of oxygen.

A final aëration is given the solution at the battery-storage tank. The arrangement here is the same as that above described. A 3-in. pump is, however, used to draw solution from the bottom of the tank and elevate it above the aërating device, where it is discharged. The use of the pump provides a continual circulation of solution. At the Nova Scotia mill it is figured that the consumption of cyanide is greatly lowered by the use of these aërating devices and the extraction is doubtles improved in like measure.

Tonopah Slime Treatment.—In the Desert mill of the Tonopah Mining Co., at Millers, Nev., the slime treatment costs 30c. per ton more than that of the sands. Ore is crushed in a 4-lb. cyanide solution which is brought up to 6 lb. in the final treatment. The sand treatment requires 14 days. Slimes are agitated 70 hr., material being in the plant about 4 days. The water consumption is 120 gal. per ton of ore treated.

Adding Lime to Cyanide Solution.—At the cyanide plant of the North Star Mining Co., at Grass Valley, Calif., a simple method of adding lime to the cyanide solution is used. Enough lime is slacked in a small box to supply the mill for 12 hr. This supply is then fed into a gold pan, such as was formerly used for grinding concentrate, in six approximately equal portions, at intervals of 2 hr. The muller of the pan makes about 1 r.p.m., and is driven by a small water wheel. The muller is loosely hinged to the driving spider and as it slowly rotates, it breaks up the lumps of lime on the bottom of the pan and slightly stirs the slacked lime into a thin emulsion. Water is added to the pan, the supply being regulated by a faucet, causing a constant overflow of milk of lime into the cyanide solution. The strength of the over flowing milk of lime is not constant, but nearly so. Although a considerable amount of lime is let into the pan at one time, the muller stirs up only a small quantity; and most of the coarse particles settle back to the bottom of the pan, so the solution is kept fairly uniform.

Lime Emulsion Feeder.—A convenient and automatic scheme for feeding lime to a cyanide mill where it is desired to add a continuous stream to a battery feed or discharge is illustrated in Fig. 192. Provide two conical-bottomed wood or iron tanks, each to have a capacity of 12 hours' supply of lime when diluted, 10 to 1, with water. The point of each tank is fitted with a valve which is opened at intervals for short periods, being held shut at other times by a spring as shown. A tiny jet of air in the point of the tank will maintain the lime emulsion at a constant

consistency. The lime should be slaked in one tank (with hot water, of course, since this method gives much the best results), and then diluted while the other tank supplies the plant. The ¼-in. screen is provided in the cone to retain contaminating rock, which is in nearly all lime, and which cannot be discharged by the valve. Slaked lime, even if nearly dry, is provokingly difficult to feed in the required small quantities, as it arches and packs and fails to keep the feeder charged.

At the Tom Reed mill, Oatman, Ariz., lime emulsion is added to the pulp as it flows from tube mills to thickeners, the amount being determined by alkalinity tests on the mill solution.

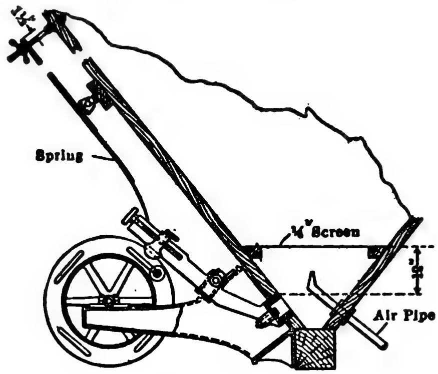


FIG. 192.—LINE EMULSION FEEDER.

Methods of Sand Treatment Compared.—At a meeting of the Institution of Mining and Metallurgy, H. A. White compared the systems of sand treatment in use at the Homestake in the Black Hills and the Princess Estate on the Rand. In the first place the separation of only 40% of slime against about 63% of —20-mesh material in final pulp seems extremely low, he said, when it is considered that the cost of slime treatment is lower, and the extraction higher, than on the sand plant. Again, the practice of running water to waste both in filling and after first cyanide solution has been applied has frequently been proved dangerous. Compared with this, the method of the Princess Estate, using a Caldecott sand filter table, with the sand free from slime obtained from diaphragm cones by double washing, and a short-circuit of 0.03% cyanide solution rarely rising above 0.3 dwt. in gold value, has obvious advantages, says Mr. White. The following representative figures are given: Slime treated, 64.37%; sand treated, 35.63%; sand residue, 0.23 dwt.; slime residue, 0.149 dwt. The final drainings on this sand average less than 0.05 dwt., which figure, Mr. White thinks, would not be attained by the system of returning gold to the vats as advocated at the Homestake.

Cyanide Treatment of Concentrates with Mill Tailings (By R. E. Tremeroux).—At the North Star mine, Nevada county, Calif., the concentrates are ground in an Abbe tube mill and run in with the tailings from one of the two 40-stamp mills. The object in doing this is to reduce the amount of fine grinding otherwise entailed. The concentrates only are ground. The stamp-mill tailings, along with the concentrates, are classified in Merrill classifiers, about 55% going to the sand tanks and 45% to the slime settlers. The slimes are agitated in 0.03% cyanide solution, and the solution extracted by Oliver slime filters. The sands are leached in 120-ton tanks. Forty tons of 0.1% solution are run through; then 125 tons of filter solution (from the slime filter); then barren solution from the Merrill leaf precipitate presses until the effluent solution shows only a trace of gold. The strong solution is run into one gold tank and the wash solution into two other gold tanks. The precipitation is done in Merrill leaf presses. Two presses are used, having a capacity of 200 tons each in 24 hr. The value of the concentrates averages $40 per ton; stamp-mill tailings, $1.80 per ton. After the concentrates are added, the slimes have a value of $4 and the sands $2.90 per ton. The tailings from the cyanide plant average about $0.30 a ton, showing an extraction of over 90%. During 24 hr. 130 tons of mill tailings and 5 tons of concentrates are used. In precipitating, 40 lb. of zinc dust are used per day. The cleanup from the presses averages 400 to 500 lb. of dry precipitate per month, valued at $20 to $30 per pound.

Treating Concentrates at Lluvia de Oro.—Concentrate worth $1000 per ton is reduced to a value of $300 per ton before being packed to the railway from the Lluvia de Oro mine in Chihuahua, Mex., by simply grinding in pans with cyanide and mercury. This is done to remove as much of the free gold from the concentrate as possible, but also to remove a part of the incentive for stealing. Besides the free gold which is caught in the amalgam, most of the gold and silver which dissolves during this grinding in pans is precipitated by, and amalgamates with the mercury, part of the latter going into solution. After the charge is ground fine, the rich solution is decanted and washed from the concentrates as thoroughly as possible and conducted to the zinc boxes where its valuable content is precipitated in the usual way. The ground pulp is conducted over a slime concentrator, reserved for this special material, and thus the concentrate is materially reduced both in bulk and value before shipment on its long journey to the smeltery. The amalgam is reduced to bullion by ordinary methods.

Tube-mill Circuit at the Alaska-Treadwell Mill.—Concentrate is cyanided at the mill of the Alaska Treadwell Gold Mining Co., at Douglas Island, Alaska. The material is brought to the plant in cars running by gravity into revolving tipples which invert them and drop the contents

into round, conical-bottom steel bins. The bins are 15 ft. in diameter and the bottoms have a 55-deg. angle. The concentrate is sampled while in the car, using a long ship-auger. Unslaked limed is added to each of the cars after leaving the tipple, forming a bed for the concentrate and also aiding in the discharge of the material. The concentrate in the bin is kept covered with water which prevents its oxidation. Tightly fitting gates control the bottom of the bin, the contents of which are sluiced directly into a Dorr classifier, the sluicing medium being the oversize return. The classifier makes 24 strokes per min. Slime from the classifier is led to the Pachuca tanks for treatment and the coarse material is delivered into a 5- × 22-ft. Abbé-type tube mill which is equipped with a spiral feeder. Corrugated sectional lining is used in the tube, which makes 27 r.p.m. The finely ground product from the tube mill is delivered into another Dorr classifier from which the slime is taken to treatment in the Pachuca tanks and the sand delivered into the foot of

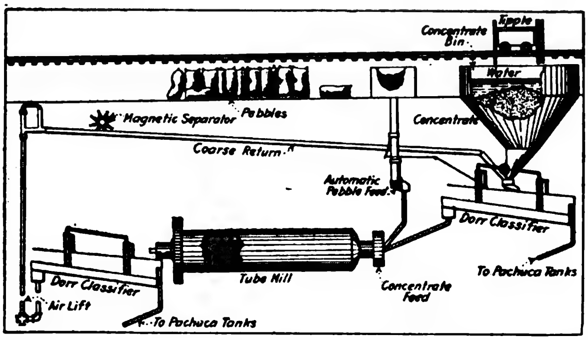


FIG. 193.—OUTLINE OF TUBE-MILL CIRCUIT AT ALASKA-TREADWELL MILL.

an air lift. The sand is lifted to a point high enough to enable it to flow back to the first classifier by gravity, and is delivered into an open launder. Accumulated iron is removed from the coarse pulp passing through the launder by means of a magnetic device which is described elsewhere in this volume. The return of this coarse oversize is used to sluice the concentrate from the bin into the initial classifier. Pebbles are automatically fed into the spiral feeder through a pipe leading from the pebble bin. The pebbles are fed, a few at a time, into the feeder and are controlled by the movement of the first Dorr classifier. The whole circuit is illustrated in Fig. 193.

Continuous Agitation in Pachuca Tanks.—In a discussion of M. H. Kuryla's paper before the Mexican Institute of Mining and Metallurgy on "Continuous Pachuca Tank Agitation at the Esperanza Mill," A. Grothe proposed an improvement in the mechanical connections between the Pachuca tanks, as shown in Fig. 194. This connection avoids

the numerous curves in the pipes of the original Esperanza installation and tends to eliminate the clogging of the pipe by sediment, which would reduce the useful section of pipe. Mr. Grothe claims that with a much smaller head, the discharge through a 4-in. pipe should answer the purpose, if no obstruction existed. The pipes have an inclination of 60 deg. and a moderate velocity of pulp keeps them clean. The connection with a flexible pipe outside the tank nullifies the effect of vibration and expansion.

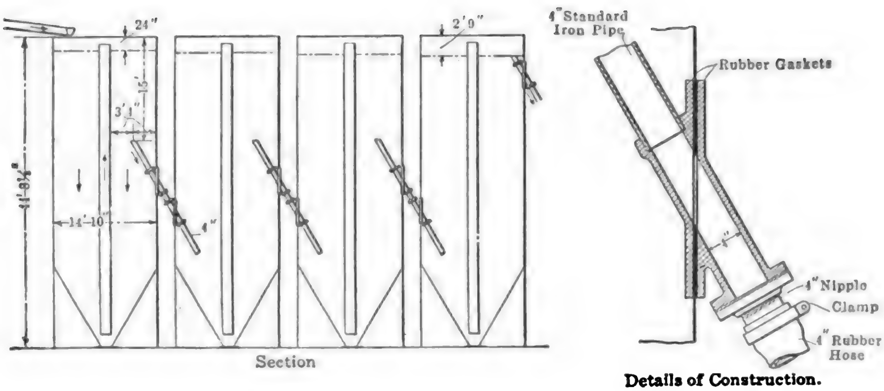


FIG. 194.—CONNECTIONS PROPOSED FOR CONTINUOUS AGITATION IN PACHUCA. TANKS.

Piping for Continuous Agitation (By H. R. Conklin).—I devised the layout of piping for continuous agitation, herein described and illustrated in Fig. 195, having in view the simplest satisfactory design. Five tanks, each 24 ft. in diameter and 32 ft. deep, with conical bottoms, were to be piped for continuous agitation, without interrupting their use for intermittent agitation, with settling and decantation.

A straight line of 4-in. pipe, with grade of 8 in. in 24 ft., was put through all five tanks, the first tank having the pipe at entrance resting on its top and at exit, 8 in. below the top. The fifth tank has the pipe at exit 40 in. below the top. The work on the tanks consisted in cutting two 4-in. holes in each tank at proper elevations and boring holes for flange bolts. Connections through the sides of the tanks were made by flanges on each side of the tank sheet using gaskets of old rubber belting. On the pipe in each tank, a threaded tee, pointing down, was placed about 5 in. from the entrance side of the tank for discharging the pulp into the tank. A gate valve was placed next to prevent any pulp from passing along the transfer pipe.

The pulp is discharged from the tank through a slot in top of the transfer pipe, about 1½ in. wide and 12 in. long. This slot is closed by a hard-

wood plug, fitted with a handle of suitable length so that it can be pushed into place from the platform on top of the tanks. This wood plug should not project far inside of the pipe, as it would hinder the flow of pulp in the pipe. The downward-pointing tees are closed, when so desired, by pipe plugs screwed in. These 4-in. pipe plugs, and the wooden plugs are both attached to the tank platform by loose chains, to prevent workmen dropping them into the tanks. To bypass any tank, it is simply necessary to screw in the pipe plug, open the gate valve and plug the discharge slot. Air agitation produces sufficient elevation of the pulp

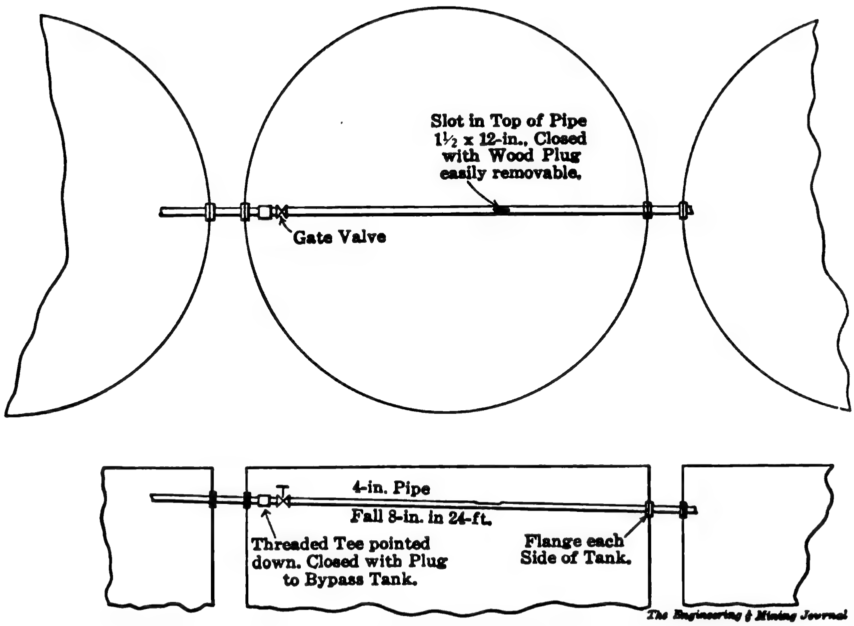


FIG. 195.—PIPING FOR CONTINUOUS AGITATION AT LLUVIA DE ORO.

in the middle of the tank to cause a continuous sampling out through the discharge slot, while a continuous supply enters through the threaded tee. The level of the pulp in the tank takes care of itself. Any tank can be bypassed and pumped out for repairs or other reason, and no difficulty is experienced in bypassing three or four tanks at the same time. The transfer pipe can be easily cleaned of any settled pulp by air or water hose passed through any of the slotted discharges. By placing both plugs in any tank, thus bypassing it, it can be used for agitation and decanting, if desired, and the change from one system to the other causes no delay.

Saving Power on Paddle Agitators.—At the Florence-Goldfield mill, by the simple device of hanging a piece of track iron from one blade

of the stirrer used in the pulp-agitation tank, the power consumption was found to be reduced one-third. The piece of iron is suspended from the paddle blade by two chain links so that as the paddle revolves, the iron drags over the surface of the settled slimes, leveling it off so the paddle blades do not scrape along. The play allowed by the links keeps the iron from having to dig deeply. It is merely the weight of the dragged iron that presses against the settled pulp. It has been found that by using this device no trouble is experienced from stirrer arms getting broken on account of the excessive strain from digging packed, settled slimes.

Wright-Jaentsch Slime Agitator.—Economical agitation of slime is an important part of cyanide treatment and the great number of types

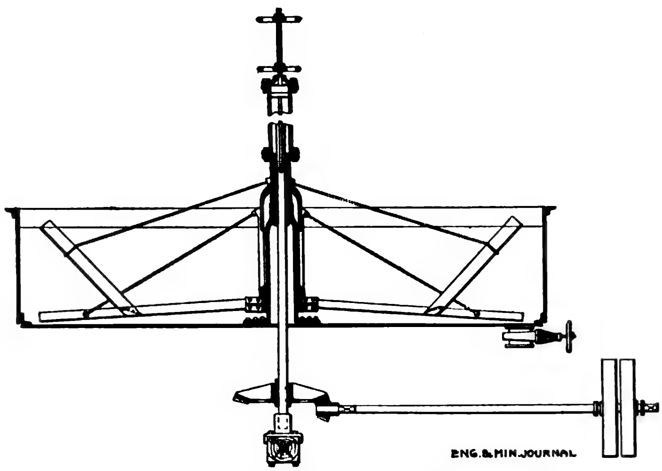


FIG. 196.—CENTRIFUGAL SLIME AGITATOR.

of agitators in use testifies to the difference of opinion as to superiority In Fig. 196 the details of the Wright-Jaentsch device, as used at the Great Boulder Perseverance mine, in Western Australia, are shown, as described by H. B. Wright in the *Monthly Journal* of the Chamber of Mines of Western Australia, June 30, 1913.

The operating principle is centrifugal force, the speed of the inclined-pipe members causing a suction in the bottom of the pipes and a discharge from the top, a speed of about 350 ft. per min. having proved most satisfactory from all points of view. The pipes should have an inclination of 45° from the horizontal, anything less being subject to settling.

In trials with this agitator in competition with simple mechanical agitators, it is claimed that a material increase of extraction was obtained.

Purifying Air for Agitating Pulp.—At a number of cyanide plants in the United States, the compressed air that is to be used for aërating the

sand or slime, or in agitating the pulp, is purified by filtration before use. Several types of filters are used; at the Homestake plants the air is passed through a filter-press made up of several cells similar in all respects to the cells of the Merrill presses used in the treatment of slime. At the Alaska-Treadwell concentrate-cyanide plant, the apparatus described by W. P. Lass (*Bull.*, A. I. M. E., February, 1912), and shown in Fig. 197, is used.

Cylinder oil or the products of its combustion or decomposition, which are introduced into the air in the cylinders of the compressors, are the impurities that should be removed as completely as possible before the air is used for agitating. The Homestake presses do this effectively by the filter cloths retaining the oil and other solid impurities contained in the air.

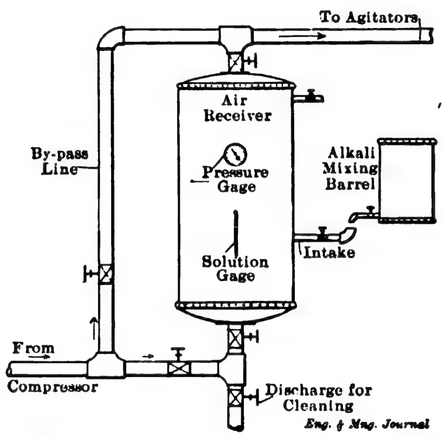


FIG. 197.—ALASKA-TREADWELL AIR PURIFIER.

The apparatus herein illustrated has a further advantage in that carbonic acid is also removed by caustic soda or milk of lime. The removal of this acid is accompanied by a decrease in the consumption of cyanide, for it is a well-known fact that carbonic acid decomposes potassium and sodium cyanides, and even in the presence of an abundance of protective alkali some decomposition by this acid may take place.

Clarifying Cyanide Solutions (By F. H. Wetherald).—Methods of clarifying solutions previous to precipitation vary in cyanide practice. Frequently they are inefficient when an excessive flow of slimy solution to the gold tanks occurs at times. Often apparently clear solutions carry fine colloidal material and complex metallic compounds to the zinc boxes, where the effect on precipitation is notably injurious.

Vertical gravity filters which I installed in two mills so satisfactorily solved the problem of obtaining clear solutions, that a description of this

filtering arrangement may prove of interest. The filters were devised by W. E. Holderman, in connection with the Holderman process, at Manning, Utah. They consist of canvas-covered frames placed vertically in the gold tanks at intervals of about 1 ft. Each filter has a bottom connection with a common discharge pipe placed on the bottom of the tank, and since it is subject to a steady gravity pressure only, much less substantial construction is required than for vacuum filters. The vertical position not only increases the possible filtering area, as compared with a bottom filter, but also allows much of the suspended material to settle between the filters to the bottom of the tank away from the filter surface.

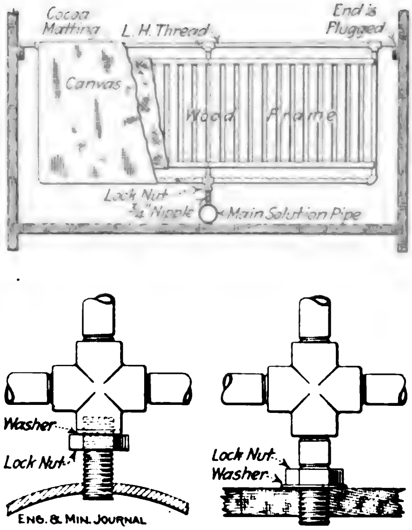


FIG. 198.—FILTER LEAF FOR CLARIFYING SOLUTIONS.

Wooden frames, with 1-in. space strips, inclosed in a canvas envelope, give good service, but for deep tanks or where solutions are persistently clouded, requiring frequent cleaning of the canvas, a more substantial construction is advisable.

In round tanks the filters may correspond in length to the respective chords of the circle, or be built to the length of an inscribed rectangle. The pipe or box conveying filtered solution from the tank is then placed centrally.

A strong frame is made by using $\frac{3}{4}$-in. pipe, arranged as shown in Fig. 198. The two horizontal pipes forming the top member of the frame have a T-joint connection with the central vertical stiffening pipe. These horizontal pipes are cut with right- and left-hand threads in order to draw the frame tightly together. The two horizontal pipes forming the

bottom member and the lower end of the vertical center pipe are screwed into a cross joint, the fourth or bottom opening of which is stripped of threads to receive loosely the nipple tapped into the main solution pipe. This nipple is provided with a lock nut and washer to insure a tight joint. Where a wooden box is used to convey filtered solution from the tank, the nipple is screwed into the cross, the box receiving the nipple and seating the washer. In the bottom pipe $\frac{3}{16}$-in. holes are drilled to provide means of exit for the solution. The inner wooden space frame is covered with coco matting and the complete frame inclosed in the canvas envelope, which is rolled and sewed at the top and about the protruding end of the cross.

At one plant, the canvas at the end of two weeks was covered with very light colloidal slime $\frac{1}{16}$ in. thick, although the solutions had always appeared clear. The filters were cleaned as follows: Solution in one of the two tanks was drawn off to a point one-third of the height of the filter, and the canvas scrubbed in the remaining solution with a long-handled brush. The muddy solution was then allowed to settle 1 hr., and the tank again placed in commission. The total time required for the operation was 3 hr. The accumulated slime was sluiced out of the tank twice a year.

In another case the tank was drained once a week, and the filter leaves cleaned by sluicing with a small high-pressure stream, the slime being pumped to a thickener. For shallow tanks 8-oz. canvas is satisfactory. Fig. 198 shows a filter leaf installation and discharge-pipe connections for either a pipe or a wooden box.

The Nahl Intermittent Slime Decanter (By Arthur C. Nahl).—In Fig. 199 is shown a slime decanter, used instead of canvas leaves by the Progreso Mining Co., Triunfo, Baja California, in the treatment of its low-grade oxidized silver ores. A 60-deg. cone is constructed inside of a wooden tank. The inside surface of the cone is constructed of brick and mortar finished with a thin surface of cement. Radial reinforcing walls 1 ft. thick extend from the inside of the cone to the sides of the tank, and the space between these walls and the tank is filled with sand, or jig tailings.

Scrapers balanced on both sides are supported by a frame which runs on wheels on a track around the upper edge of the tank, and are steadied by a pin at the bottom. The slimes enter the center and settle on the sides of the cone, whence the coagulated slimes are assisted to the bottom by the slow-moving scrapers without any stirring action. The clear solution overflows at the periphery of the tank. The alkalinity of the solution must be carefully watched to obtain the best results.

The frame carries a foam fence 3 ft. high above the solution, and running down to 6 or 8 in. below the surface. It is made of $\frac{1}{32}$-

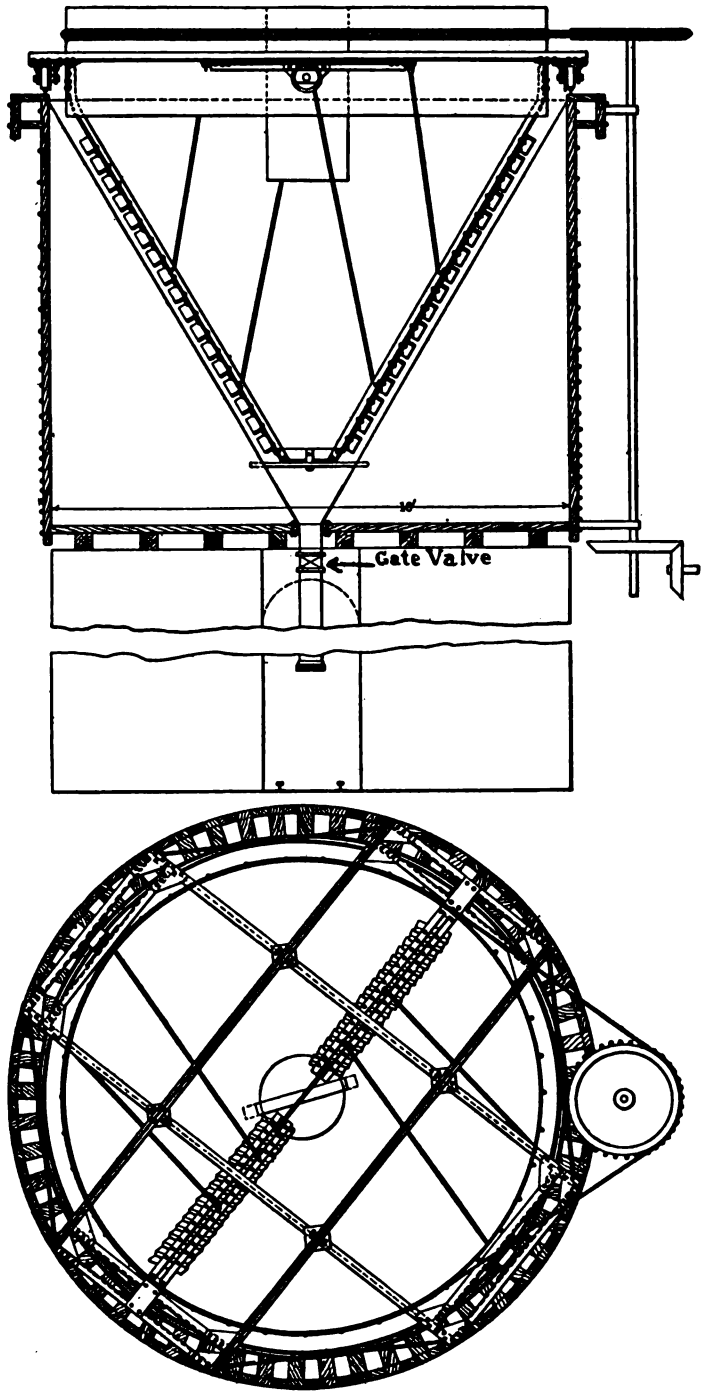


FIG. 199.—NAHL INTERMITTENT SLIME DECANTER.

in. steel, and is riveted to the inside of the sprocket band. The foam accumulates in this and dries, and drops in gobs to the bottom when it is too heavy to float. In this way it does not interfere with the clear overflow by being carried over the edge by winds.

The sprocket is a homemade affair. The pins are riveted in a band of $\frac{1}{4} \times 3$-in. iron. The pins are riveted about a foot apart and set exactly to engage a No. 67 sprocket chain. It is revolved by a 10-in. sprocket. A smaller driving sprocket is actually used than is shown in the diagram and the whole scraping apparatus is revolved by a worm gear at the bottom instead of the bevel gear as shown in the drawing. The chain is guided on the driven and also the driving sprockets by small rollers not shown in the drawing.

The slimes accumulate and are compressed by their own weight in the bottom discharge pipe, and are discharged intermittently through an appropriate gate at the bottom of the pipe into a car. If it is inconvenient to revolve the scrapers by mechanical power, a small boy can turn them around slowly every 2 or 3 hr. or so by a crank fastened to the worm shaft.

This decanter was designed by me with the idea of doing away with canvas filter leaves in treating low-grade base silver ores, and also to meet the requirements of out-of-the-way mining camps where other filtering devices might be costly. Possibly some other managers may find this settler effective.

A Slime Filter Frame.[1]—In the slime plant of the Talisman mine, Karangahake, N. Z., there are two filters of the fixed-frame type, each set consisting of 31 leaves, each 9 ft. long and 4 ft. 9 in. wide, equal to 2650 sq. ft. of filtering surface. The frames were spaced $7\frac{1}{2}$ in. between centers, but 8 in. is found to be more convenient. They are connected at each end to a 4-in. manifold leading to the vacuum pump.

A gage glass is fitted in the connection to one of these manifolds, so that a defective cloth is readily noticed. Any leaf may be disconnected and removed without interfering with the rest. The details of the frames are shown in Fig. 200. Black corrugated iron with 1-in. corrugations forms the support for the cloth and is held by a frame formed of $\frac{3}{4}$-in. piping, having a $\frac{1}{2}$-in. slot cut longitudinally to receive it. The tanks containing the filter frames are 20 ft. long, 10 ft. wide and 7 ft. deep, and are of $\frac{3}{16}$-in. steel plate, stiffened by vertical and diagonal braces of angle iron $4 \times 4 \times \frac{3}{8}$ in. The bottom consists of two square pyramidal hoppers 7 ft. deep, formed of $\frac{1}{4}$-in. steel plate and stiffened with a horizontal rib of $3 \times 3 \times \frac{1}{4}$-in. angle iron placed half way down. The filter frames are

[1] Excerpt from an article entitled "Mining and Ore Treatment at the Talisman Mine, Karangahake, N. Z.," by Arthur Jarman; *Proceedings*, Australasian Institute of Mining Engineers, September, 1911.

supported at either end by a $3 \times 3 \times \frac{1}{2}$ angle iron. The weight of the tank is supported by a $5 \times 5 \times \frac{1}{2}$-in. angle iron riveted on the side and resting on an 18×7-in. H-girder supported on cast-iron columns. At the ends of the tank similar angle irons rest upon 12×5-in. H-girders, the ends of which are supported by the ends of the 18×7-in. girders.

The capacity of the filter tank is a little more than 60 long tons. A full charge comes up to 6 in. from the top of the tank. Agitation is effected by compressed air, admitted from $\frac{1}{2}$-in. pipes, two near the bottom of each hopper. The slime is prevented from impinging on the filters, while the tanks are being filled, by a $\frac{3}{16}$-in. baffle plate.

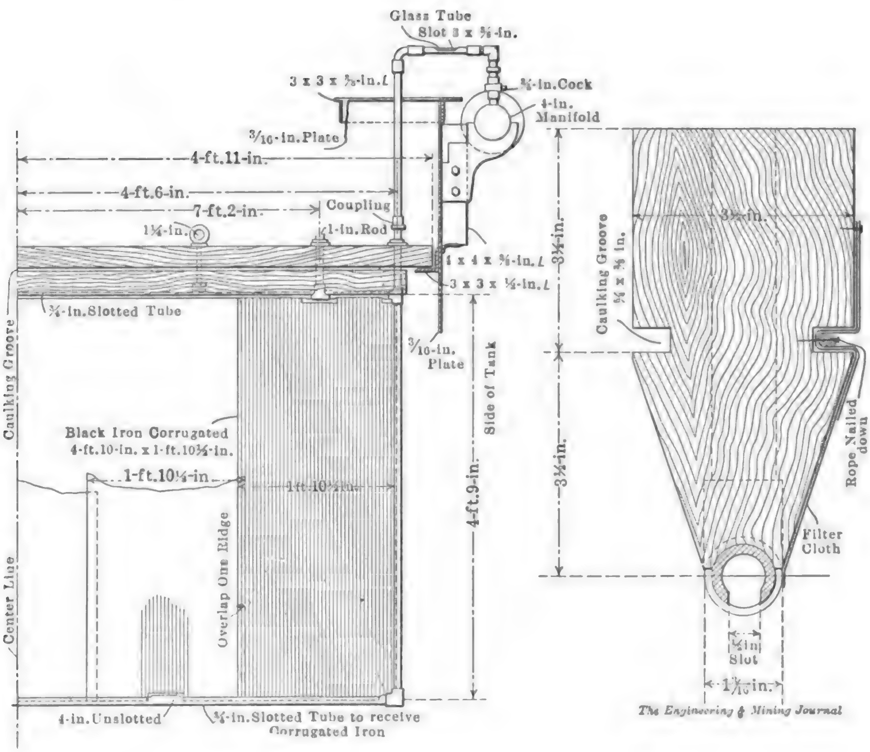


FIG. 200.—SLIME FILTER LEAF USED AT TALISMAN MILL, KARANGAHAKE, N. Z.

There are two vacuum pumps to work to a net lift of 40 ft. They are wet, double acting, belt driven, and either side of the cylinder can be disconnected. The cylinder is 18 in. diameter by 18 in. stroke, and at 50 r.p.m. a vacuum of 23 to 25 in. of mercury is maintained. An hour at this vacuum suffices to build up a $\frac{3}{4}$- to $1\frac{1}{4}$-in. cake, when the cloths are new, and a 1-in. cake is equivalent to about 10 tons of dry slime. Old cloths work slowly, owing to the formation of scale. During the forma-

tion of the cake the solution is pumped to a 15 × 15 × 4-ft. settling tank, from which it goes to the weak-solution settler, prior to entering the zinc boxes. The slime remaining in the tank after the cake is formed is pumped out into a storage puddler vat, 22 ft. diameter by 5 ft. deep. Weak wash solution is run in from one of the two weak-wash storage vats, and washing is continued from 1½ to 2 hr. according to the grade of the material under treatment. The remaining wash solution is then pumped back to its tank, and the vacuum continued for another 15 min. to drain the cake as completely as possible when it carries 18 to 25% of solution. During this 15 min. the sluicing gates at the bottom of the hoppers are opened, and the cake is sampled by taking a grab from one side of every other leaf. Vacuum is then released and the cake assisted to fall by the use of a wooden spatula and by hosing. This residue assays about 5s. per ton. Discharging takes a little more than 10 min. with new cloths, but old cloths require an hour or more.

A New Filter Frame.[1]—The filter frame at present being used in Western Australia is shown in Fig. 201. The trouble with all the earlier frames was that the cakes cracked on exposure and consequently fell off. Practically all frames are now built of 1-in. steam pipe and are covered with 16-oz. duck, with coco matting between.

The filter cloths are sewed with vertical seams 3 in. apart, this spacing having been found to be the best. Undoubtedly a great part of the filtered solution passes through the stitch holes instead of the cloths, and it has been found that bags with seams 6 in. apart do not take cake so quickly as those with 3-in. spacing. Bags with seams 1 in. apart have been tried, but these did not form the cake so quickly as the 3-in., the reason being that the close stitching drew the cloth tight and compressed the coco mat so that little space was left for the solution to pass through.

The weight of the pipe, frame and cakes is supported by two beams of special section. The detail of these beams is important. The beam at first used, Fig. 1 (referring to Fig. 201) had many disadvantages. The wooden cleats were not strong enough to hold the cloth, and the weight of the cake. The cloth also had no support where it joined the beam, thus causing cracks to form there on exposure. It is also obvious that if the cloths were allowed to stretch, the cake would crack at the top. Another difficulty was that the cake formed on the cloth, where it was stretched over the beam and could not afterward be dislodged by the air discharge, thus necessitating constant hand cleanings. The air pressure also caused rapid wear of the cloths, for when the air was turned on a great strain was put on the vertical seams at the top of the

[1] Excerpt from an article in the *Monthly Journal* of the Chamber of Mines of Western Australia.

bag, and it was only a matter of a few charges before holes appeared, allowing the slime to leak into the filtered solution.

All these disadvantages have been overcome in the improved frame, Fig. 2. In this frame the support of the cloth is satisfactory, as it is clamped securely in the split beam. No cloth is exposed on the outside of the beams, consequently no cake can form on it. The shape of the edge *A* is designed so that it just presses on the cloth, but not so tightly as to prevent the passage of compressed air. The vertical seams come just up to this ledge, and it is, therefore, impossible for them to rip.

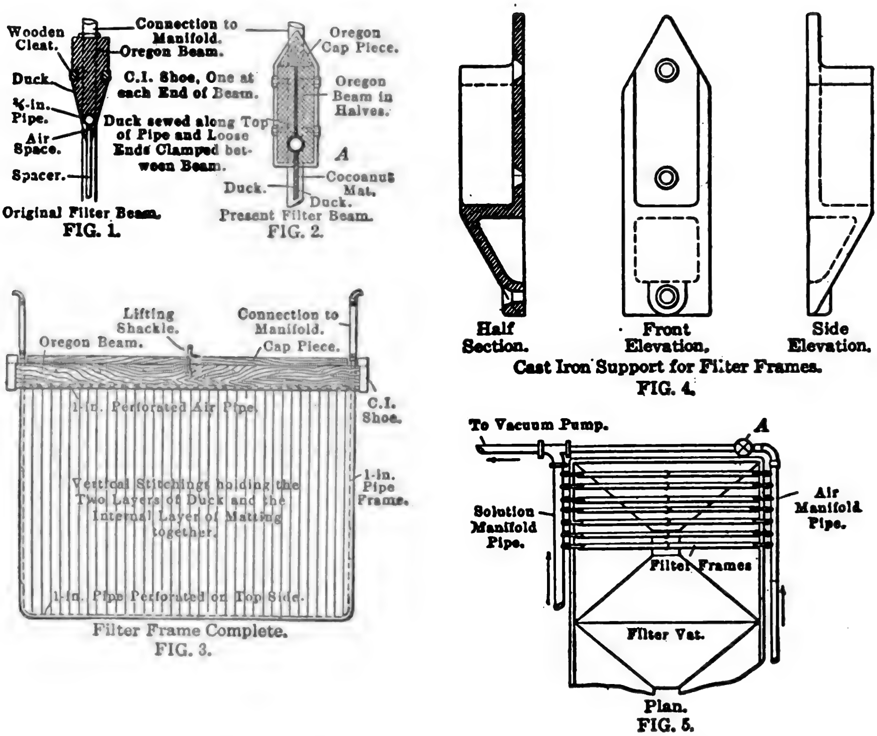


FIG. 201.—DETAILS OF FILTER-FRAME CONSTRUCTION.

Frames are now usually made of a uniform size (Fig. 3) 9 ft. wide by 4 ft. 9 in. deep; this permits the use of a covering of duck, 72 in. wide. The capacity of the plant is adjusted, not by the size, but by the number of the frames employed. Taking into account the handling of the frames for renewal, and the time occupied in filling and discharging the filter vats, this size of frame has been proved, from practical experience, to be the most efficient; deeper frames take considerably longer to discharge, owing to the choking of the cakes.

If a plant is working on slime that will form a cake less than 1½ in. thick, piping of smaller size than 1 in. may be used for building the frame; but with cakes above this weight nothing less than 1-in. pipe will stand the shock of the falling cakes. Pipes are liable to break where they join the beam, owing to the leverage exerted on them when the falling cakes spread the frames at the bottom. It is advisable not to fix the frames in the filter vat too securely, as a slight motion of the beams decreases the shock to the frame. Figs. 2, 3, and 4 show the latest design for holding the frames in position. The cast-iron shoes are securely bolted to the frames and have a good footing in the cast-iron sockets bolted to the filter-vat side. This arrangement allows for slight rocking of the frame; it will withstand considerable wear and tear, and the frames will always hang plumb.

Various devices have been tried for obtaining the necessary spaces for the flow of the solution between the cloths of the frame. Wooden laths were first used, one being placed between each 3-in. seam; these were fairly satisfactory, but they did not give the support to the cloth that the coco mat did, and the cakes were, therefore, more liable to crack during exposure. Ripple iron, *i.e.*, galvanized iron with small corrugations, as used at the Waihi mine, New Zealand, was also tried, but as this had to be used without vertical seams, trouble was experienced in discharging the cakes; when the air was turned on the cake was stretched, but not discharged.

With very free-filtering slime, *i.e.*, slime that will form a cake of 2 in. or more in 20 min., it has been found that the cakes formed thicker at the bottom than the top. The reason for this has been traced to the fact that the filter mat is not sufficiently porous to allow the solution to be drawn off as quickly as it passes through the cloth. This difficulty has been obviated by using the top pipe of the frame, as well as the bottom one, for drawing off the solution. This method is illustrated in Fig. 5, which may be described briefly as follows: In taking cake or washing, both the solution and air cocks on the frame, as well as the valve A, are left open, thus enabling each cock to draw from approximately half a frame. When the cakes are ready to discharge, valve A and the air valves on the frame are closed; compressed air is then admitted to the other end of the air manifold, and the air cocks are opened one at a time to discharge the cakes. The same result could be obtained by using larger pipes for the frames and a larger solution manifold, but this would add needless weight to the frames. This arrangement is also useful as an addition to an old plant, where, by reason of the internal deposition of lime, the pipes become too small for the flow of solution.

The Caldecott Sand Filter Table.—The Caldecott sand filter table is one of the appliances used in connection with the Caldecott system of

continuous sand collection. This table takes the place of the collecting vats that were formerly required in all cyanide plants on the Rand. The pulp issuing from the batteries undergoes a preliminary treatment in the Caldecott, diaphragm, cone classifier, in which a separation of sand from slime is made. The underflow, a thick sand containing from 26 to 30% moisture, is carried by a small additional stream of water through a launder and is discharged upon the sand filter table, where the water is removed prior to treatment by cyanide solution. One of the advantages of the table is that half an hour after the pulp leaves the battery, after having been crushed in water so as to enable flowing it over amalgamation plates, the excess water can be removed and the cyanidation of pulp be started.

The underflow from the first diaphragm cone is led to a secondary cone or cones with the addition of sufficient water to enable the resultant pulp to flow readily through launders set at a grade of not less than 10%. The overflow of the secondary cones carries with it about 80% of the slime left in the underflow of the primary cone, so that a final product containing about 1 to 1½% slime remains for treatment at the filter plant. As the efficiency and durability of the filter depends largely upon the freedom of the sand from slime, it will be seen that efficient working of the

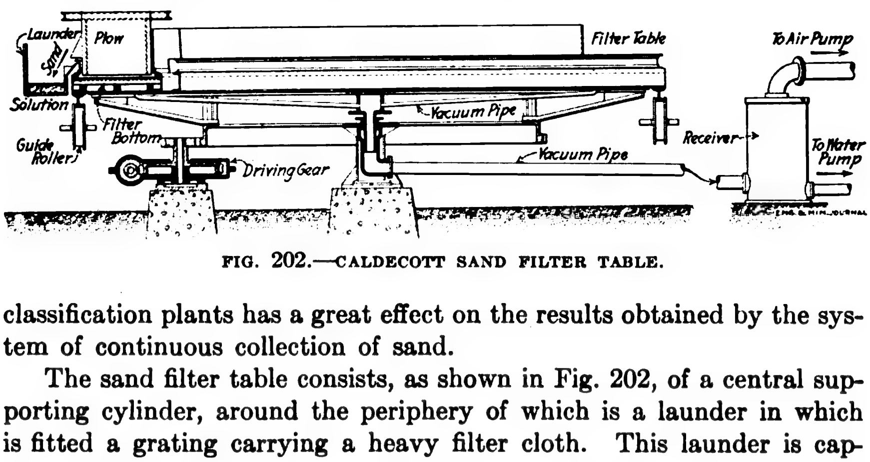


FIG. 202.—CALDECOTT SAND FILTER TABLE.

classification plants has a great effect on the results obtained by the system of continuous collection of sand.

The sand filter table consists, as shown in Fig. 202, of a central supporting cylinder, around the periphery of which is a launder in which is fitted a grating carrying a heavy filter cloth. This launder is capable of being revolved at a speed of one revolution every 3 min. it being mounted upon rollers, as shown, for that purpose. The bottom and sides of the launder must be air-tight, and should be well coated with bitumen, applied hot, after the usual calking has been done. The grating is made of strong screening, about 9 mesh, with No. 12 or 14 wire, and is supported by pieces of wood set radially on edge and beveled so as to reduce the filtering area as little as possible. On the grating is laid coco matting bound at the edges and calked well into the sides

of the launder. As coco matting shrinks on becoming wet it must be cut to fit the launder and placed in position while moist, otherwise it will be liable to pull the calking away from the sides of the launder. A strip of jute about 6 in. wide should be placed between the matting and the grating at the outer and inner edges to render the filter less porous at these places. This is necessary to obtain an even flow of air through the filter, as the layer of pulp from the classifiers is usually thicker at the middle than at the sides. To prevent slime reaching the matting and choking it, strips of unbleached calico are cut to fit the launder and laid on the matting without being fastened.

The sand is removed from the revolving table by a stationary plow placed in a slanting direction across the launder, and made so that it may be raised or lowered a few inches when desired. Care must be taken, however, that it does not approach closer than ½ in. to the filter cloth, and that the variation in the distance between the filter cloth and the plow does not exceed ⅛ in. during a revolution of the table. The launder is attached to a center column on which it revolves, and is driven by gears actuated by a 5-hp. electric motor. The center column is hollow, and is connected to a receiver to which a vacuum pump is connected at its upper and a water pump to its lower portion. Pipes are placed to connect the bottom of the launder and the column so that the space between the bottom of the launder and the filter cloth may be partly exhausted as is shown in the accompanying illustration.

The pulp from the classifiers is distributed at a point about 3 ft. behind the plow, so that the sand must make almost a complete revolution on the table before being removed. In actual working the plow is raised so as to leave a space of 1¼ in. between its lower edge and the filter cloths, and the table is put in motion. The pulp from the classifiers being directed on to the filter and the pumps started, the superfluous moisture is drawn through the filter to the receiver, from which the water pump returns it to the mill supply. The space between the plow and the filter cloths becomes filled with a bed of sand which forms the working bed and prevents slime being drawn into the matting. This bed, and the calico, must be renewed at least once every 24 hr. The operation is readily performed, and in actual practice takes less than an hour.

The vacuum, as shown by a reliable gage, must not be allowed to rise higher than about 10 in. of mercury. If this does occur, the working bed should be carefully raked, when the vacuum will drop once more to normal, provided the bed has not become choked with slime. A table of this type, 20 ft. in diameter with a 2½-ft. filtering bed, is capable of handling 1 ton per min. and will regularly reduce the moisture in 800 tons of sand per 24 hr. from 30 to 12 or 14%. The power required to

drive a plant where two filters are used, handling about 1600 tons of sand per day, is about 59 hp., as follows: Two filter tables at 5 hp. each, 10 hp.; two water pumps at 3 hp. each, 6 hp.; one sand pump elevating 40 ft., 40 hp.; one pump supplying solution to gland of sand pumps, 3 hp.; a total of 59 hp. Separate receivers, vacuum, and water pumps, are required for each table installed, and should be arranged so that any receiver may be connected with any table, to permit repairs being made without stopping the plant. In Fig. 203 a diagrammatic scheme of how this filter table may be used in the treatment of sand is indicated by a section of a flow sheet.

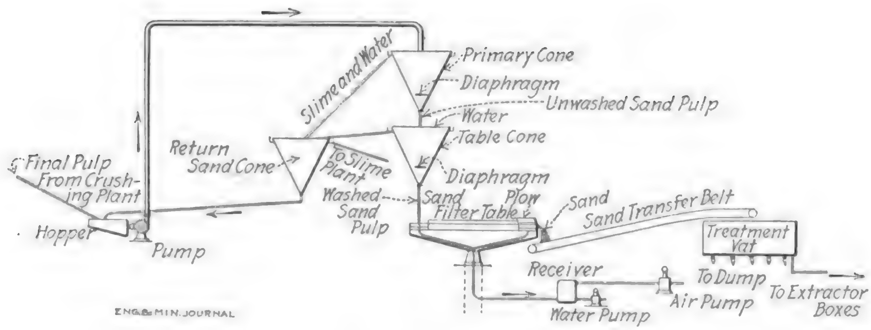


FIG. 203.—FLOW SHEET, SHOWING POSITION OF SAND TABLE.

These sand tables, each handling sand from over 2000 tons of ore per day or about 13% of the total Rand tonnage, are used at the Simmer Deep-Jupiter, Simmer & Jack; Cason (section of East Rand Proprietary Mines); Angelo (section of East Rand Proprietary Mines); and are to be installed at the Princess and at the City & Suburban mines.

Wear on Filter Leaves.—In transferring vacuum-filter leaves into the wash tanks the greatest wear comes on the corners, which are liable to scrape against the sides of the tanks. As soon as the canvas is worn through in one place the leaves are, of course, rendered useless. Heavy canvas patches sewed over the corners of the leaves, will take up this wear and greatly increase the life of the leaves. At the Montana-Tonopah cyanide plant solutions are kept at an alkalinity of $\frac{3}{4}$ lb. CaO to the ton in order to facilitate settlement of fine slimes. This results in the formation of an excessive amount of $CaSO_4$ and necessitates washing three leaves from each box of 62 every 24 hr. A wash of 2% HCl is used. As there is only 2 in. of clearance in the filter boxes, the ends and corners of the leaves soon became worn through, before the simple scheme of reinforcing with corner patches of extra thick canvas was adopted.

Renewing Filter Leaves.—A number of the mills using vacuum filters of the Butters type find it convenient and cheap to rebuild the worn

leaves. The usual practice is to discard the expensive coco matting and substitute grooved wooden strips. These strips may vary in lateral dimension but extend the full height of the leaf. Strips, 2 in. wide with $\frac{1}{4} \times \frac{1}{4}$-in. grooves on both sides, are satisfactory. The canvas filter cover is stitched between the strips so that each one slips in a separate pocket. Filter leaves constructed in this manner are thoroughly satisfactory, wear well and can be built at a reasonable cost at the works.

Cleaning Filter Leaves.—The ordinary practice at cyanide plants is to clean the filter leaves of lime about once in four weeks by removing them from the filter tank and leaving them to stand in a special tank filled with a 10% solution of muriatic acid. At one plant the cake of washed pulp is removed from the leaves by running spent cyanide solution into them from a tank placed so as to have a head of 20 ft. and holding 4 tons of solution. In cleaning the filter pores of lime this tank is filled once a week with hot water from the compressor jackets and a 125-lb. carboy of muriatic acid poured into the tank. This hot hydrochloric-acid solution is then allowed to run down into the filter leaves in the tank, and owing to the strong solvent action of the hot solution, the lime is effectively removed from the pores of the canvas. The acid bath is then followed by a water wash to remove the acid out of the leaves.

Acid Treating Filter Leaves (By C. H. Fox).—A device for assisting the acid wash of filter leaves is shown in Fig. 204. It is in use at the West End mill, at Tonopah, Nev. The leaves are placed into the acid tank and connected to a 3-in. pipe by means of the usual filter couplings. This pipe leads to the bottom of a drum 8 ft. high, made from 9-in. casing. A 2-in. pipe connects the top of the drum with the main vacuum pipe of the filter. A stop and drain valve, placed in the pipe above the drum, automatically controls the vacuum and air inlet. This valve is fitted with a lever. A weight is suspended from one end and a galvanized-iron tank from the other. This tank is fitted with two pieces of $\frac{1}{2}$-in. rubber hose, one at each end. The upper hose leads to the top of the drum and connects with the 2-in. pipe between the stop and drain cocks and the drum. The lower one is fitted with a valve and connects with the drum about 18 in. above the bottom. After the filter leaves have been placed into the acid bath and connected to the pipe leading to the bottom of the drum, the valve in the pipe between the top of the drum and the filter vacuum pipe is opened. The acid is then drawn through the filter leaf and up into the drum. The level of the acid in the drum is for a time shown in the gage glass. As the acid rises in the drum some flows through the lower hose into the galvanized-iron tank. After a level has been reached at which the weight of the acid in the tank exceeds the

weight on the other end of the lever, the tank drops and closes the valve at the top of the drum. The vacuum is shut off and air admitted through the air inlet *A*. The acid returns to the acid tank by gravity. When nearly all the acid has returned to the acid tank, the weight of the galvanized-iron tank has decreased and the excess weight at the other end causes it to rise. The vacuum is once more turned on and the cycle repeated. Practice has shown no saving of time is effected, but a much more thorough washing of the leaves is obtained.

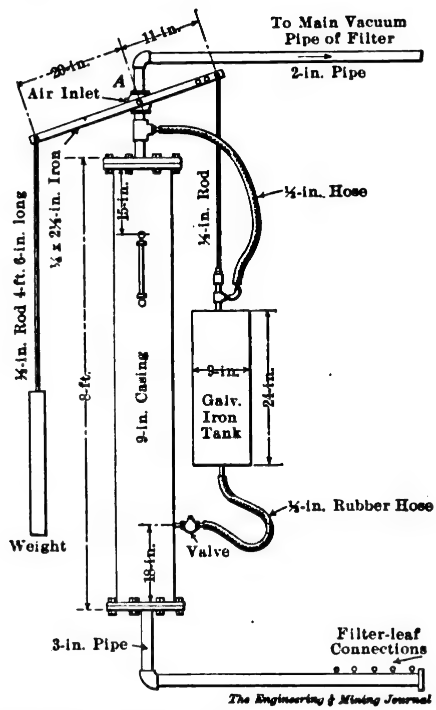


FIG. 204.—AUXILIARY APPARATUS FOR ACID-TREATING FILTER LEAVES.

Method of Cleaning Screens and Filter Leaves.—A recent invention of Maynard J. Trott, Colorado Springs, Colo., (U. S. patent No. 1,052,191) assigned to the Dorr Cyanide Machinery Co., Denver, Colo., provides a method of cleaning filter leaves, and screens of stamps, chilean mills, etc. The method consists essentially of drying the filtering fabrics, screens, etc., so that the coatings of lime and slime become brittle, and then subjecting them to the abrasive action of sand violently projected against the surface. The inventor claims that the adherent matter is speedily and thoroughly removed from both inner and outer surfaces of screen or fabrics by the sand blast, but the force of the latter is not sufficient to penetrate the fabric.

Winding the Oliver Filter (By Henry B. Kaeding).—The winding of the Oliver filter drum, either on installation or on replacement of filter cloth, is an operation that consumes considerable time and may be unnecessarily drawn out unless steps are taken to facilitate the operation. Assuming that the drum makes a revolution in 8 min., it will require 77 hr. to place the wire in position, unless the speed be increased. The manufacturers advise an arrangement of pulleys somewhat similar to my own idea, whereby this time may be cut to considerably less; but the best arrangement of pulleys and countershafting will be found outlined in Fig. 205.

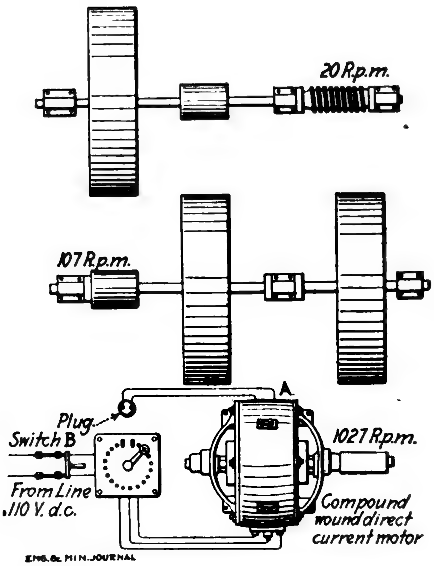


FIG. 205.—ARRANGEMENT FOR WINDING THE OLIVER FILTER.

In assembling the filter I have found it necessary to use the greatest care in making all joints air-tight. The staves that form the body of the filter should be laid in place with the greatest care, and when the ends of the piping are screwed into the staves, these ends should be first hot-tarred and a wisp of oakum packing should be wound about the pipe so that the lock-nut may screw down on it. When all pipes and horizontal strips are in place a liberal application of hot tar (asphaltum) should be given to both sides of the periphery of the drum, running it in around the pipe from both sides where they pass through the wood.

It is not possible to exercise too much care in the endeavor to have the joints air-tight and the compartments isolated hermetically from each other; and even after painstaking work there will be found many places where air will enter and aid in destroying the vacuum. Particularly annoy-

ing is the passing of the blowoff air from compartment No. 24 back under the filter cloth to the other compartments that are under vacuum at the time. Placing the screens so that no fold or wrinkle stands up above the plane of the rest of the screen offers no particular difficulty, the idea being to avoid high spots that would wear holes in the canvas. The filter cloths may then be put on and the calking ropes driven down and edges tarred.

An examination of the drawing will show a method of arranging the motor, pulleys and countershafts, where the motor is assumed to run at 1027 r.p.m. and the filter drum to take 8 min. per revolution, the number of teeth in the wormwheel being assumed to be 160. The motor drive pulley, a rawhide pulley, 10- or 12-in. face and 5-in. diameter, is belted, either with or without idler, to a 48-in. pulley on the main countershaft. It is preferable to use for this belt an endless one of leather without an idler, setting the motor on rails that the belt may be kept tight. If an idler is necessary let it be of the type that fastens to the motor frame, causing the belt to cling better to the drive pulley by bringing more surface in contact with the face. At the other end of the main countershaft will be found a 9-in. pulley that belts to a 48-in. pulley on the wormshaft of the filter, thus giving the shaft a speed of 20 r.p.m. Should there not be room for this size pulley on the wormshaft furnished by the manufacturer a longer shaft may be used, allowing the pulley to clear the end of the filter box; or a slight reduction in the size of both pulleys may be made.

A 48-in. pulley on the countershaft is placed opposite a 9-in. pulley on the wormshaft, without boxing between; this permits of throwing the belt off one set and on the other without trouble, the filter then being belted for winding and the wormshaft having a speed of 570 r.p.m. and the filter drum a speed of 3½ r.p.m. If it is desired to expedite matters still further an arrangement such as is shown at the motor at *A* may be made use of. At any convenient point such as *A*, the shunt field wire may be cut between coils and the two ends brought out and connected to an ordinary porcelain wall socket located near the starting box. Into this socket screw a fuse plug that has been prepared by placing No. 12 copper wire in place of the fuse wire. The result is that when the plug is screwed in till it forms part of the circuit, the motor is a compound-wound machine of its regular type of winding and of constant speed, but when the plug is removed the motor becomes a series-wound motor and the speed is doubled. A switch could be used in place of the plug but is not as foolproof, offering a temptation to the ignorant to pull it out and investigate which would cause the filter to double its speed.

To operate this device conveniently, start the motor with the plug in and bring all up to full speed with the starting box; then pull the main-line switch *B*, unscrew plug, and when the lever of the starting

box has sprung back, throw in main-line switch *B* and immediately catch up the speed to full with the starting box. If this is neatly done there is little if any diminution of speed in changing the motor from compound- to series-wound. The machinery may also be started on the series motor with the plug permanently out, but care must be exercised to start slowly and get up speed gradually and the starting box is almost certain to overheat during the operation. It is preferable to start with the compound winding and switch off to the series as soon as full speed is attained.

The utility of this method is obvious, the filter drum making nearly 8 r.p.m., or a revolution in 8 sec.; at this rate the entire winding of wire can be put on in 1 hr. and 20 min., exclusive of stops. In practice, however, it will be found to take much more time owing to delays in getting up speed, in slowing down for stops, in repairing broken wire and in stopping for stapling. It is advisable to stop every six or eight turns for stapling unless the operator is sufficiently skillful to drive his staples while the drum is in motion.

Discharge Door for Filter Vats.—In the cyanide plants of Western Australia water is so scarce that every economy must be practised to

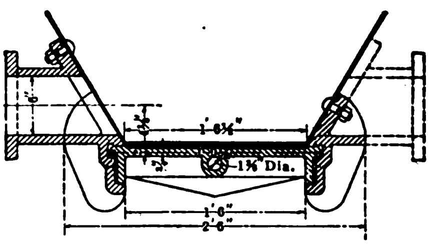


FIG. 206.—CROSS-SECTION OF HORIZONTAL FILTER-VAT DISCHARGE DOOR.

husband the supply. Sufficient water is not available at some of the mills to discharge the filter vats by sluicing as is the usual practice in the United States.

Where it is required to discharge the residue into the pond in the form of a pulp as thick as can be handled, the hopper-bottom filter vats are situated directly over a residue vat provided with a mechanical stirrer, into which the residue from the filter vats can be discharged and then be mixed with a minimum quantity of water.

The type of door used in the bottom of the filter vats is shown in Figs. 206 and 207. A cast-iron flange is riveted to the bottom of the hopper of the filter vat. The frame of the door, which is also made of cast iron, is bolted to the flange. The door itself is slightly wedge-shaped and slides in grooves so cut that when the door is closed it presses upward against the flange closing the opening so tightly that no leakage occurs.

When the door is opened there is no obstruction to prevent the free passage of the thick sludge through the orifice. The door spindle is bolted to the door casting and is cut with treble square thread of 1½-in. pitch so that quick opening is possible. The door is operated by a 24-in. handwheel.

The time taken to discharge the residue is about 10 min. and the discharge door may be closed and the cycle of operations in the filter vat be resumed while the stirrers are mixing the thick pulp in the vat below, preliminary to discharging to the pond by means of belt or other conveyors.

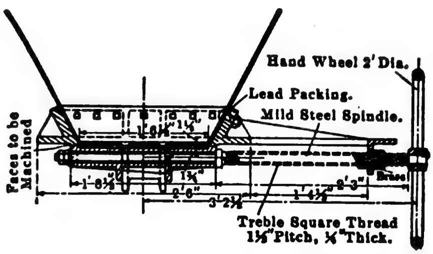


FIG. 207.—LONGITUDINAL SECTION OF HORIZONTAL FILTER-VAT DISCHARGE DOOR.

PRECIPITATION OF PRECIOUS METALS FROM SOLUTION

Handling Cyanide Precipitate at Lluvia de Oro (By H. R. Conklin). —At the mill of the Lluvia de Oro Gold Mining Co., Lluvia de Oro, Chihuahua, Mex., zinc-shavings precipitation is giving way to the zinc-dust method. The latter has proved much more satisfactory, since experience has enabled the correction of faulty operation. The substitution of a centrifugal for a triplex plunger pump has also been of advantage.

The emulsion of zinc dust and solution produced in the centrifugal pump is filtered through a press, as in the usual practice, but additional capacity was required, and a system of sacks is now used, which is proving satisfactory. After many experiments the sacks are being made double, with the outside sack of 8-oz. duck, and the inner sack of plain cotton sheeting. The inner sack is made larger than the outer sack, so that it does all the filtering, while the outer sack takes all the pressure. Sacks are made tapering, and by using the patterns as shown in Fig. 208, are cut from the cloth without any waste. Inner sacks are sewed on a regular household sewing machine, and outer sacks on a heavy belt-stitching machine, with heavy linen cord. Both are sewed by doubling over the edge and sewing through the four thicknesses. Inner sacks are placed inside the duck sacks by means of two long slender sticks similar to a pair of shears, and are carefully spread so that when solution is turned on, the

inner sack immediately fills out against the outer sack, without any spaces between.

Each of two sets of sacks is about equivalent to a large press in capacity, and consists of 44 sacks, suspended from ¾-in. pipes, which are tapped into 4-in. pipe headers and spaced at 8-in. centers. The two 4-in. pipes are suspended about 8 ft. above the floor, with centers about 10 in. apart, and each sack connection to the header consists of a short ¾-in. nipple next to the header, with a valve, nipple, union, nipple and sleeve below. Sacks are tied at the top with heavy linen twine, the inner one first, and the outer over it, the sleeve on the lower end of the pipe serving as a shoulder to prevent the sack slipping off when under pressure.

Underneath each set of sacks is placed a flat pan or launder on a slight grade, which catches all the dirt and is piped to the barren-solution tank. Each sack holds about 50 lb. of precipitate when taken off, and costs less than 50c.

In operation, two, three or four sacks are replaced each day, and precipitation is not stopped for cleaning up. The cycle of a sack is about 11 days. Beginning when its valve is opened, the flow of solution is

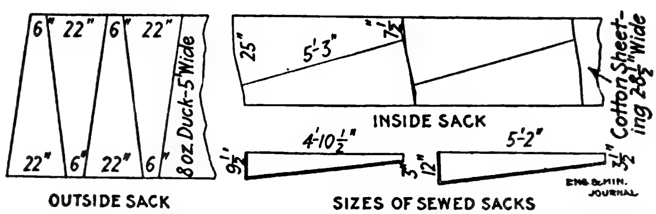


FIG. 208.—PATTERNS FOR MAKING FILTER BAGS FOR CYANIDE PRECIPITATE AT LLUVIA DE ORO.

rapid, but after the first day, becomes much less. After its valve is closed, it is allowed to hang one day and dry, and is then disconnected at the union and taken down. The binding twine is cut, nipple washed, and a new sack tied on, ready for work. The filled sack is set in the strong room for a few days to dry farther, and is then cut open, when the precipitate is in a hard cake. The sacks are burned in the electric furnace when melting precipitate, so that there is no loss.

Regulation of the amount of solution precipitated is obtained by using any desired number of sacks. Contact of the precipitate with air has no effect, and if zinc dust is properly fed, all precipitation has occurred before filtering, so that the efforts of some managers to prevent air entering the precipitate press seem entirely unnecessary. Either in a press or in these sacks, the precipitate builds up from the bottom, and an excess of zinc, once placed in the filter, has no further appreciable effect in precipitation.

The only advantage of the press is that the precipitate is locked and safe against fire, but sacks could just as well be placed in a fireproof, locked room, or isolated building, and in every way are more satisfactory than the press.

Zinc-dust Precipitation at Brakpan Mill.—The precipitation equipment at the Brakpan Mines plant, on the Rand, consists of three Merrill presses with belt feeders. Trouble was experienced at first (*Bull.* 101, I. M. M.) in obtaining a regular feed of zinc dust, owing to the poorly constructed feeding mechanism. This difficulty was eventually overcome and the belt system formed a satisfactory feeder. It was also found necessary to supply the presses with exceedingly clear solution. Slightly turbid solutions that would not have any effect on the precipitation in zinc boxes, caused the cloths to choke, and caused a rapid rise in pressure in the press, necessitating bypassing and resting of the press. It was also found advantageous to keep the cyanide strength higher than would be the case with zinc boxes. In spite of the troubles that were experienced at first, it came to be admitted that the presses possessed several advantages when compared with the zinc-shaving system. Less time and labor were spent on the cleanup. There was no gold left in the plant after the cleanup. The space occupied by the plant is smaller than that occupied by zinc boxes. The richer gold slime left after acid treatment and calcining resulted in decreased time and expense in smelting. There were fewer samples to be assayed, and, finally, the gold in the press was safe from theft. There still exists the variability in the nature of the zinc dust supplied; some consignments being much more efficient than others.

Barrels as Zinc Boxes (By John Tyssowski).—Cheap and efficient vats for the precipitation of gold from cyanide solution with zinc shavings can be made from empty whisky barrels. These are, unfortunately, only too easily to be had around most mining camps. To make the barrels ready for this use it is only necessary to saw them off about 3 in. from the top and bore a hole for a connecting pipe at a point about an inch below the rim. A satisfactory arrangement is to have three lines of barrels arranged so that the solution is divided and passes through three barrels before being discharged. A screen false bottom should be put in about half way the depth of each barrel. The solution runs off the top of one barrel, passing through an iron pipe, usually 1- or 2-in. diameter, to the next barrel into which the pipe is turned down, terminating just above the screen. Solution passing off the top of this barrel is piped into the next in the same manner. Short zinc and precipitate fall through the screen and accumulate in the bottom of the barrel. A series of such precipitating barrels are used in the Florence mill, at Goldfield, Nev. It is claimed that three lines of barrels are as efficient as two

boxes, each having eight 14-in. compartments. At the Florence mill, the head solution passing to the precipitating barrels assays from $2.80 to $4, the tails only 24c. per ton.

Screen Trays for Zinc Boxes.—More than half the expense of the ordinary cyanide plant cleanup can be avoided by using screen trays, the scheme first adopted on a large scale at Dos Estrellas, Mex. Gold and silver, when precipitated on zinc, are ordinarily in a state of minute subdivision, except where copper is deposited with them. If, then, the precipitate and short zinc accumulated in a cleanup be scrubbed and rubbed vigorously over a 60-mesh screen in the cleanup tank, the precipitated metal will pass the screen and the short zinc will remain on it. The gold and silver which pass the screen can be melted directly without further acid or other treatment, beyond drying and fluxing.

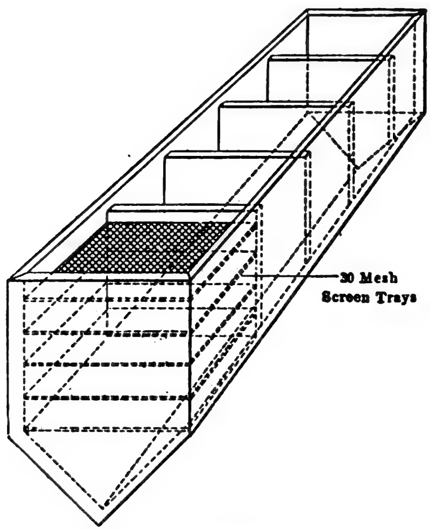


FIG. 209.—SCREEN TRAY.

Then, if one or more of the compartments of a zinc box be provided with 30-mesh screen trays as shown in Fig. 209, the mass of short zinc can be spread in thin layers on these trays. It will remain pervious to solution, and will precipitate gold and silver. Strong solution in preference to weak should be passed through these trays, thus securing full value from the zinc, reclaiming all the gold and silver which adheres to the short zinc and avoiding entirely the sulphuric-acid treatment.

Vacuum Filter for Zinc-box Slimes (By Lyon Smith).—A vacuum filter for use in small leaching plants, where it is not expedient to install a filter press, is shown in Fig. 210. The filter box is constructed of 2-in. boards, preferably redwood or fir, which are held tightly together by

drawbolts. The inside wooden frame is for the support of the iron screen over which is placed an 8-oz. canvas filter cloth. The canvas is cut about 5 in. larger than the screen and is held in place by rope calking around the edges. An ordinary distillate or gasoline drum makes an excellent storage tank. The slimes from the cleanup vat are delivered

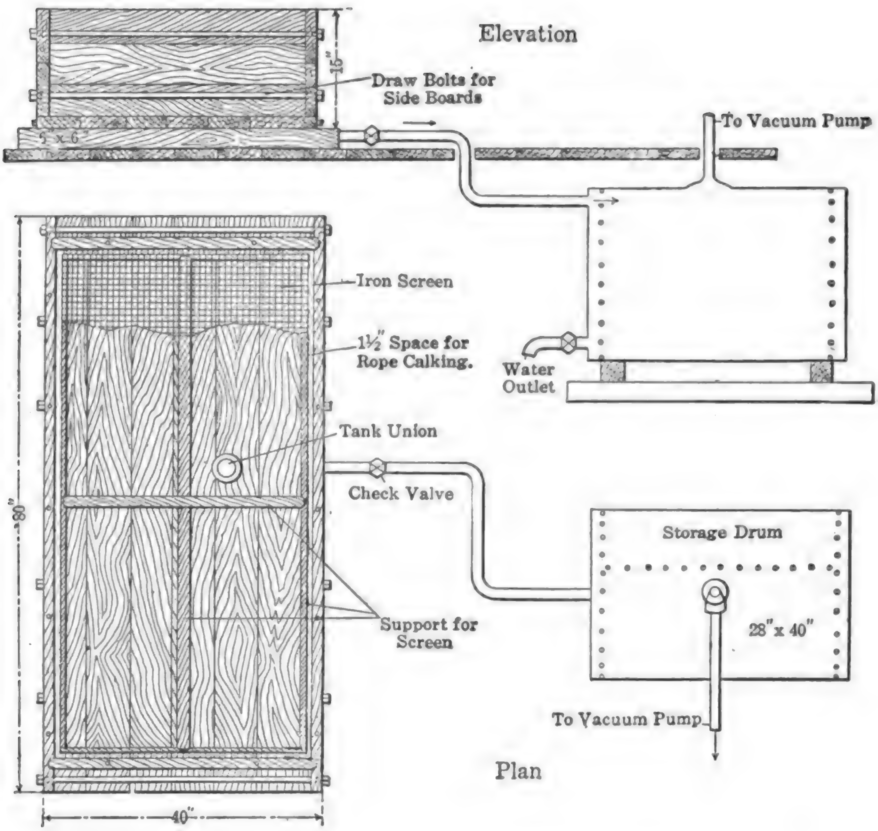


FIG. 210.—VACUUM FILTER FOR ZINC-BOX SLIMES.

to the filter and the clear water passes to the storage drum, from which it is drawn off to waste, or, if desired, it may be conducted back into the system. The slimes are reduced to about 30 to 33% moisture and are then removed to the drier. A cleanup from a 4-ton settling tank, from which about 3 tons of clear solution are first pumped off, can be made in from 4 to 5 hours.

Zinc-dust Feeders (By A. B. Parsons).—The simple but satisfactory method in use at the Goldfield Consolidated mill and a number of other Western cyanide plants for the zinc-dust precipitation of cyanide

solutions is illustrated in Fig. 211. In the Consolidated mill there are three precipitating sump tanks, one for strong and two for weak solutions. By simply changing valves the zinc dust can be switched to any of the three tanks, solution being run into the others while the contents of one are being pumped through the presses. In each tank there is a float, attached to which is a cord that passes, as shown in the sketch, about the pulley that advances the rubber belt carrying the dust. The belt is thus advanced as the level of the solution lowers in the tank and at a rate depending upon the adjustment made. The number of tons per inch of solution in the tank, or what is the same thing, per equivalent length of belt, is known and the desired amount of zinc per ton of solution is spread along the belt. Fresh water mixes with the zinc in a funnel-like arrangement and flushes it through a rubber hose to the intake of the main precipitating line for the particular tank being pumped. Two triplex pumps are used to pump the solution to the Merrill presses in the

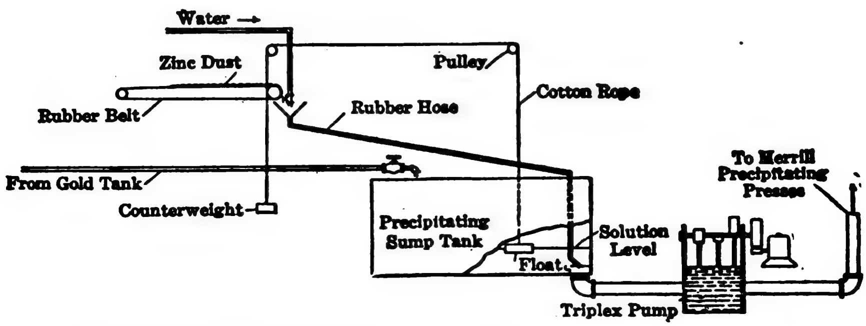


FIG. 211.—ZINC-DUST FEEDING AND PRECIPITATING ARRANGEMENT.

refinery at the upper end of the mill. The precipitation is largely effected during passage through about 400 ft. of pipe. Not only does the device require little attention, but it assures an exceptionally uniform introduction of the dust in exactly the required quantity.

In the Montana-Tonopah mill at Tonopah, Nev., where the same method of introducing zinc dust is employed, it has been found advisable to introduce compressed air along with water in the funnel into which the belt discharges the dust. A better emulsion of the dust is thus obtained and more complete precipitation assisted thereby. The accompanying drawing only indicates the arrangement of the cone in which the zinc dust is emulsified. A $\frac{3}{4}$-in. water line is turned down into the cone, terminating close to its bottom. The outlet pipe from the cone connects to the precipitating-pipe inlet. It is a 1-in. line and is provided with a valve at a point between the cone and where connection is made to the fresh-water system. This water pipe is also controlled by a valve. The fresh water is only used to flush out the system.

An Improved Zinc-dust Feeder (By Claude T. Rice).—It is important when using the zinc-dust method of precipitation from cyanide solution to feed the zinc dust to the solutions evenly, for, if the feed is not even, too much gold is precipitated in the press itself. In the old method of precipitating the solution by means of zinc dust, as practised for years at the Consolidated Mercur mill, at Mercur, Utah, the zinc dust was poured in large quantity into a vat where it was violently agitated for a time with the solution, but in a few hours the bulk of the zinc dust was in the presses and the main precipitation had to be done there. This resulted in a large excess of zinc in the precipitate so that it became necessary to cut down the precipitate by means of sulphuric acid before refining. Moreover, there is an oxidation of the zinc dust in the press which results in a higher consumption of zinc.

Mr. Merrill, when he adopted zinc-dust precipitation, devised the method of feeding the zinc dust to the solutions by a belt that has become almost an integral part of the process, as most of the zinc-dust precipitation plants in the country have been installed by him. The great drawback to the belt system is that the length of belt that it is practical to use in feeding the zinc dust is limited and so the zinc dust must be piled on the belt in a thick layer. As a result, the zinc dust is not fed continuously to the solution as theory requires, but instead at times does not feed to the emulsifier for several minutes; especially when the air is a little damp, it banks up and does not give away at the end until its angle of repose has been surpassed. Then the zinc dust starts and feeds quickly until it has assumed again a slope that is equal to the angle of repose that limits motion, quite a different thing on account of friction and inertia from that which has to be surpassed in order to start motion again in the zinc dust. Of course, the emulsifier, which consists of a cone with an air jet for keeping the zinc dust in suspension, acts somewhat as a trap to equalize the feed of the zinc dust to the solution going to the suction pipe of the pump, but still the feed is intermittent and probably not so effective as when the zinc dust drops continuously into the emulsifier. At the Goldfield Consolidated mill, where the Merrill belt feed is used for supplying the emulsifier with zinc, observation has shown that the zinc will at times hang up so that not a particle of dust is fed to the emulsifier for as long as five minutes.

Ordinarily, in the Merrill method of feed, the belt is driven from floats in the gold tanks, but sometimes this is not convenient as the gold tanks often have small capacity. Such is the case at the Central mill at Grass Valley, Calif., where the belt is driven by means of a ratchet-actuated gear wheel driven from the solution pump that feeds the presses. This ratchet method is necessary in order to step down the high speed of the

pump to the slow feed of the zinc-dust belt. By means of varying the throw of the ratchet the feed of the zinc is adjusted.

A minor drawback of the belt feed is that, in case water should drop on the zinc dust on the belt and there should be any decomposition of the zinc, a hole will be burnt in the belt and in case no one is near, the

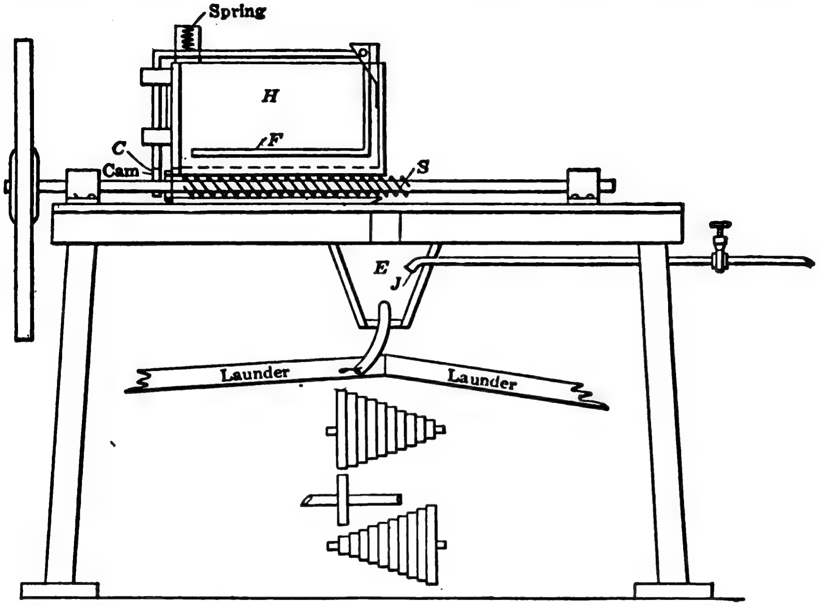


FIG. 212.—ZINC-DUST FEEDER, MONTANA-TONOPAH MILL.

whole belt will burn up. Such an accident happened at the Central mill, where fortunately someone happened to be nearby at the time so that the fire was put out before it had greatly damaged the belt. Moreover, the spreading out of the zinc dust evenly upon the belt takes time. These objections are all avoided in the zinc-dust feeder devised by B. A. Bosqui, superintendent at the Montana-Tonopah mill, which gives a positively continuous feed of zinc dust to the emulsifier. It has resulted in the saving of 30% in the zinc consumption over the method of feeding the zinc by hand first used at the mill.

In the feeder devised by Mr. Bosqui, the charge of zinc dust to be fed to the solution is stored in a hopper. In this hopper *H*, referring to Fig. 212, there is a feeding arm *F* which is moved up and down so as to keep the zinc dust from packing in the hopper and so not feeding down through the slot in the pipe that envelops the screw feed below. This arm is the end of a belt-crank lever system that is raised by a cam mounted on the screw shaft of the feeder, while the return is effected by a spring. The feeder proper consists of a slotted pipe 1.5 in. in diameter in which rotates

a screw that feeds the zinc dust ahead in the pipe and drops it in a continuous stream into the emulsifier *E*. This screw is turned from an old piece of 1.5-in. shafting and has threads cut in it about half an inch deep and pitched two per inch.

The emulsifier consists of a small box with a hose attached to a short nipple as a discharge so that the feed of zinc dust can easily be turned into whatever one of the three launders leading to the different gold tanks that it is desired. The emulsification is effected by a spray of barren solution that issues from a jet *J*., pointed downward at an angle of 45 deg. so that it hits the bottom of the emulsifier box at the point where the zinc dust strikes as it falls from the screw feed. Owing to the considerable force of the spray, as the barren solution issues from the jet under a pressure of 40 lb. per sq. in., and the fact that the jet strikes the bottom at an angle of 45 deg., a good emulsification is obtained and at less expense than when a jet of air is used in a cone.

The emulsion of zinc dust and solution then flows through the launder to a small pipe which takes the emulsion to a small sump box in the gold tank about 12 in. square and 9 in. deep with which the suction pipe of the pump supplying the press at the head of the mill is connected. In this way an effective emulsification of the zinc is obtained and an intimate mixture of individual zinc-dust particles with the solution is effected.

The speed of the feed of zinc dust to the emulsifier is regulated by means of a set of cone pulleys driven from the main shaft that also drives the solution pump feeding the zinc press. By means of these cone pulleys the speed of the big driving wheel is changed through a range of 3 to 15 r.p.m. as each step on the cones is equivalent to a change of one revolution in the speed of the big wheel. This driving wheel is made large so that a steady drive to the screw is obtained.

The wheel's speed is changed only by the superintendent, who varies the speed according to the assay of the tail solution from the zinc press. Generally after a cleanup the pump is speeded up so as to get an excess of zinc in the press; then gradually as the precipitate accumulates in the press the speed is cut down, watch being kept on the tail solution until there is only a small excess of zinc in the press.

At present the men keep track of the amount of zinc fed by means of a schedule on which is given the capacity of the head tanks in tons corresponding to different heights in inches of solution, and the amount of zinc fed per ton according to the number of revolutions at which the wheel rotates and the amount of drop in the head solution tank. The solution man times the wheel and gets the number of revolutions and then reads the tell-tale for that tank and gets with these data, from the schedule, the amount of zinc that has been fed. But as the amount of zinc fed per revolution of the screw varies with the atmospheric conditions, the feed

computed in this manner has to be checked frequently by using weighed amounts. Consequently, it is the intention in the future to keep the amount of feed of zinc by means of weighing the actual zinc put in the hopper of the feeder.

Automatic Zinc-dust Feeder (By James S. Colbath).—An automatic zinc-dust feeder which has been used with entire satisfaction at the El Rayo mill in Mexico is shown in Fig. 213. The use of this device lowered the zinc consumption materially over the old method of feeding by hand into a cone agitated by compressed air. I do not make any claim of excellence in this respect over any other device that will give a continuous and uniform feed. For simplicity, positive action and uni-

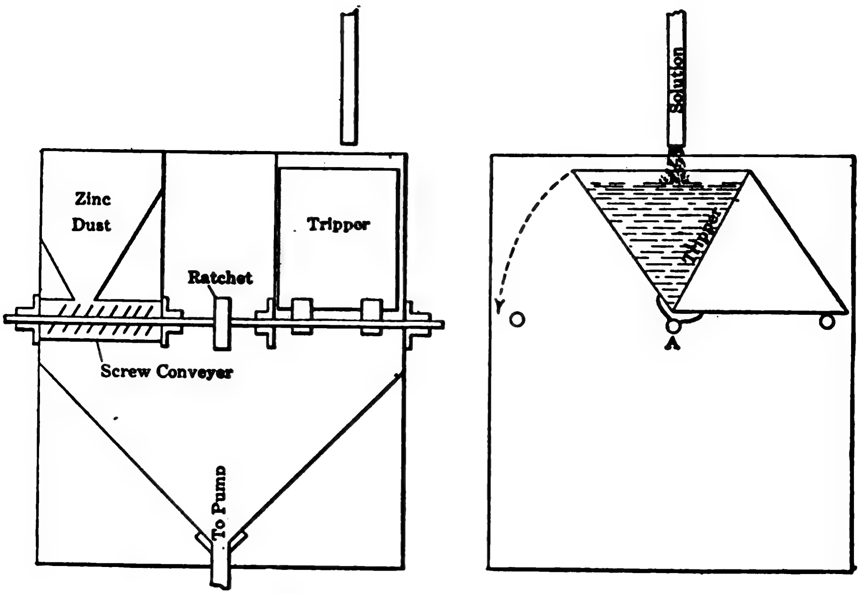


FIG. 213.—AUTOMATIC ZINC-DUST FEEDER.

formity, however, it cannot be excelled. The apparatus consists essentially of a receiver having diamond-shaped cross-section with a central partition. The hopper is mounted on a shaft *A*, and the whole caused to oscillate by flow of solution. The shaft is connected with a screw conveyor by a ratchet causing the screw to revolve only in a positive direction. The zinc drawn from a hopper by the screw mingles with solution discharged from the tripper and flows to a precipitating pump. The feed is controlled within the limit of capacity by the flow of solution. The sketch, I believe, will be clear to anyone. I would caution those attempting to construct a feeder of this kind, not to have the opening over the screw too constricted, as there is a tendency for the zinc to arch. Also, the

tripper must be accurately constructed and the bumpers so placed that the center of gravity when full of solution will be on the same side of a vertical plane through the shaft as the direction in which it should rotate. The operating arrangement of this machine, with the addition of a spoon, is familiar to most millmen as an excellent pulp sampler. I am unable to give credit to the inventor. A lime feeder on the same principle, but differing materially in construction, is credited to P. S. Taverner.

Device for Handling Zinc Shavings.—The handling of zinc shavings used in precipitating gold and silver from cyanide solutions is a problem which has involved a great deal of annoyance and expense. The zinc thread, when cut, is usually piled away in a more or less tangled mass, and when it is desired for use, an appreciable expense is entailed in straightening it out and getting it ready. The great amount of handling causes a breakage of a goodly proportion of the thread, making a product that is not of much use in the boxes and which is usually wasted. A method of winding the zinc shavings in convenient form for use has been devised whereby this secondary handling may be entirely avoided, obviating both the annoyance and expense of the operation. The winding device consists, as is shown in Fig. 214, of a horizontal shaft carrying at its end a light crosspiece of wood. From the ends of the latter two short pieces,

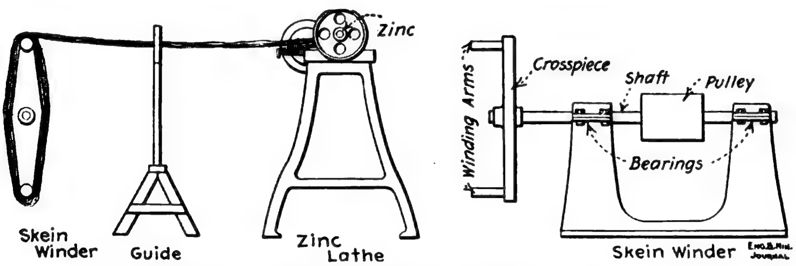


FIG. 214.—DEVICE FOR HANDLING ZINC SHAVINGS.

which may be of small pipe or wood, are fastened. The whole virtue of the idea depends on the fact that the distance between these two projecting arms is equal to the width of the compartment into which the zinc has to be packed. The shaft is moved by a pulley, the speed of travel of the arms being made to equal exactly the velocity of the zinc thread as it comes from the lathe. The result is that when one end of the zinc thread is fastened to one of the traveling arms, the shaving is wound continuously into a skein which will fit into the compartment of the zinc box without further handling. When the skein is sufficiently thick it is slipped off the arms, the thread cut and another skein started. The skeins of zinc are stored away and are ready for instant use when required without further trouble or expense. A movable guide directs the thread to the skein from any point on the lathe. The device is installed at the Liberty Bell mill,

near Telluride, Colo., where its use has been attended by economy and satisfaction. W. H. Staver, general superintendent of the Liberty Bell, is responsible for the idea and he suggests that the scheme might be adapted for winding skeins of various sizes by replacing the cross piece with a wooden disk having holes at various intervals into which wooden pins might be placed upon which the various lengths of skein might be wound.

A Double Tool Zinc Lathe.—At the Desert mill of the Tonopah Mining Co., a lathe is used for cutting zinc shavings, the unusual feature of which is that shavings are cut simultaneously on each side of the mandrel. The two cutting tools are fed against the zinc by a screw feed. The screw feeds, one for each of the cutting tools, are operated by pawls, driven by eccentrics that engage ratchet wheels on the screw-feed axles; in this manner the high speed of the driving shaft is stepped down. The screws make 3.5 r.p.m. and are cut with threads so spaced that a forward travel of 0.58 in. per min. is secured. On the mandrel 22 sheets of No. 9 zinc, 36 in. wide and 108 in. long are wound. The tools cut this into shavings in 58 min. so that the lathe has a capacity of 100 lb. of zinc an hour. The mandrel on the lathe is cooled by passing some of the head solution through the mandrel as it is pumped to the precipitation boxes.

VII

SMELTING

NOTES ON EQUIPMENT AND GENERAL PRACTICE

The Power Plant of the Copper Queen Smeltery.—The Douglas power plant of the Copper Queen is described by Charles Legrand, in *Bull.* 101, I. M. M. The boiler plant consists of eight 500-hp. water-tube boilers, equipped with economizers, and working at 155 lb., gage pressure. In addition, there are four 520-hp. boilers utilizing the waste heat of reverberatory furnaces situated 1300 ft. from the boiler house, generating steam at 180 lb. to allow for pressure drop. All the steam is superheated to 460° F. in a separately fired superheater, and reaches the engines at about 410° F.

The fuel used is California crude oil containing 18,400 to 18,500 B.t.u. per lb. and weighing 8.05 lb. per gal. (231 cu. in.) at 60° F. In regular operation, the heat content of the superheated steam over that of the feed-water to the economizers, is from 83 to 85% of the total heat in the fuel.

The boiler-feed pumps are electrically driven, and no feed-water heaters are used, the feed-water being taken from the condenser hotwell. The fuel-oil pumps and steam used in atomizing the fuel oil take approximately 4% of the total steam generated; the exhaust of the oil pumps is used to heat the fuel oil.

The power plant consists of 11 blowers, driven by tandem-compound engines; six of 300-cu. ft. capacity per revolution and five of 200 cu. ft. Each is piped separately to one furnace, and the quantity of air delivered is fixed by the speed of the blower, the pressure, usually 26 oz. per sq. in. adjusting itself to the condition of the furnace.

There are six blowing engines for the converters, five of 6000 cu. ft. per min. capacity and one of 12,000 cu. ft. per min. These engines have cross-compound steam cylinders and duplex air cylinders. They deliver air at 11 lb. pressure to a common main pipe; the speed of the engines is varied by an air governor to keep a constant pressure.

There are four 400-kw., direct-driven, 260-volt generators, connected to cross-compound steam engines. The current is distributed to the various departments by independent feeders equipped with integrating watt-meters, so that power used on any line can be measured. There are in use between 50 and 60 stationary motors, 7 electric cranes and 14 electric locomotives.

In addition to the direct-current generators there are two 750-kw., mixed-pressure turbines, delivering a 60-cycle, three-phase, 2300-volt, alternating current, transmitted at 44,000 volts to a mine 67 miles away.

All the engines are equipped with individual ejector condensers; they can also be connected to a common exhaust pipe, which takes the exhaust steam at approximately atmospheric pressure and delivers it to turbines; the engines so connected are then running noncondensing, and the number of engines connected to turbines is varied, according to the load. If there is more exhaust steam than is needed by the turbines, it is automatically bypassed direct to a turbine condenser, and if there is not enough exhaust steam for the turbine load, high-pressure steam is admitted automatically to a separate set of nozzles acting on the same turbine wheel as the exhaust steam. The turbine condensers are of the barometric jet type, with separate dry-air pump to maintain a better vacuum than is practicable with ejector condensers.

The circulating condenser water is cooled by a natural-draft cooling tower, and two sets of pumps are used, one for pumping from the hotwell to the top of the cooling tower, and one from the tank under the cooling tower to the condensers. The speed of the pumps is automatically regulated according to the demand for water.

The average power developed by the plant for the first 5 months of 1912 was 4700 i.hp., which required 13.65 lb. of steam per indicated horsepower-hour. The oil burned was 1.125 lb. per indicated horsepower-hour, including that (about 1% of total) for generating steam used in various parts of plant outside of power and boiler house. Of the 4700 i.hp. generated, between 6 and 6¼% are used by boiler-feed pump, economizers, scrapers, condenser-water circulating- and air-pumps, and lighting of boiler and power houses.

Results of Furnace Enlargements at the Granby Smeltery[1] (By Frank E. Lathe).—The enlargement of the furnaces at the Granby smeltery has brought about many changes at this British Columbia plant. It has been conclusively proved that, other things being equal, the tonnage smelted per square foot of tuyere area will be greater for long than for short furnaces. This is true for several reasons. As the end surfaces are the same, the enlargement having affected the sides only, the cooling surface has not increased proportionately to the area. Moreover, as the percentage of end surface has decreased, so also has the formation of accretions in that they have much less opportunity to adhere firmly to long sides than to the ends and corners.

The matte and slag will be hotter and more fluid owing to the increased flow from the furnace. As the ratio of cooling surface to tuyere area has been reduced, the amount of jacket water per ton is decreased,

[1] Excerpts from an article in *Bull.* Can. Min. Inst., June, 1910.

and with it the loss of heat, thus effecting a saving of coke. Usually it will be found that the labor required does not increase proportionately to the output. All these points are true for the lengthening of furnaces without deepening, and the Washoe smeltery at Anaconda, Mont., may be given as the most conspicuous example of success attending such enlargement. There are, however, still further and no less important advantages to be derived from the deepening of furnaces, in some instances, at least, and these will now be considered in the case of the Granby smeltery.

As a rule, the deeper the column of ore in a furnace, the less sulphur will be burnt off, resulting in a lower grade of matte, and as this requires more iron to unite with the sulphur, the slag will be somewhat more silicious. With the deepening of the Granby furnaces, the copper in the matte was decreased 5 to 10% and the iron in the slag about 1%. This meant more matte to handle, additional work for the converters, and more converter slag to be returned to the furnaces, as well as a more difficultly fusible blast-furnace slag. Here, however, the disadvantages ended.

When the matte is low grade more silicious custom ore will be used for converter lining, and although this only partly offsets the additional expense, yet the cost of converting compared with that of the initial smelting is small per ton of ore treated. Moreover, the converter slag is useful in the furnaces, and often assists materially in righting a furnace that is working badly. It is worthy of notice that however much iron is taken from the blast-furnace slag to form a low-grade matte, all is finally returned to the furnace, the only disadvantage being that some additional silica is returned with it.

The smaller the amount of sulphur burned off in a furnace, the less chance there is of the formation of metallics. This has been a marked improvement in the deepening of the Granby furnaces, especially in the case of the two furnaces with the smaller tuyeres.

The present blast-furnace slags are more silicious than formerly, as already mentioned; but the operation of the furnaces is so much more even that there is less difficulty. A good tonnage has been maintained with 47% silica slag over a period of several weeks, while before deepening there would have been difficulty in preventing the furnace from ultimately freezing with this slag. The tonnage attained per square foot of tuyère is about 10% greater than formerly.

In the old furnaces, with a lower column of ore, much of the heat necessarily extended to the top of the charge, so that the gases on leaving were highly heated. This not only caused a needless loss of sensible heat in the gases, but the high temperature to which the downtakes and the steel flue chamber were subjected necessitated frequent repairs. By a deepening of the furnace this hot gas has to pass up through a body

of cooler ore, to which it imparts a considerable portion of its heat, thus saving from 2 to 3% of coke, and at the same time decreasing the damage done to the metal flues. The labor required at the smeltery to handle the larger output is practically the same as before. The greatest saving of all, however, is found in the amount of copper that passes into the slag. The lower grade of matte, greater matte flow, hotter matter and slag, and uniform running of the furnaces all tend to a more perfect separation of matte and slag, and the danger of the formation of copper oxide is decreased. The percentage of copper in the slag has thus been lowered by 0.05%. While this quantity may not seem great at first sight, it amounts to $150,000 to $200,000 in the course of a year.

Reducing the Capacity of a Blast Furnace (By T. Kapp).—A successful experiment to reduce the capacity of an ordinary lead blast furnace was worked out at Zeehan, in Tasmania. The furnace was 120×42 in. at the level of the tuyeres and 20½ ft. in height from top of crucible to feed-floor level. On the long sides were 7 cast-iron jackets, each having a tuyere 2¾ in. in diameter. The slag tap was at its usual position in one of the short sides and the bullion siphon near the slag tap was the only one of the two siphons used, as the low-grade ore produced a small quantity of bullion. The furnace required 80 tons of ore per 24 hr. at a blast pressure of about 25 in. of water. But for a long period the supply of ore was only 50 tons per day, so that it was necessary in order to avoid intermittent smelting and to maintain a regular run of the furnace, to decrease the size of the furnace. This was done by removing two of the jackets on each of the long sides. To accomplish this a firebrick wall 13.5 in. thick was erected in the crucible parallel to the short sides of the furnace. The clear space of 20.5 in. between this wall and one short side of the furnace was packed solidly with a mixture of fine coke and clay. The short side jackets were then put up in the usual manner and connected to the remaining five side jackets. This shortened the inside length of the furnace by 34 in. The shaft was reduced in size in a similar manner. This firebrick wall, as well as the coke and clay filling, was supported by rails, which in turn rested on the girders that carried the lining of the two long sides of the shaft. These girders were also supported by two columns standing on the top of the crucible outside of and behind the jackets, on the short side opposite the slag tap. The modified furnace was in commission for about 9 months, giving quite satisfactory results. When the supply of ore had increased to a point that permitted the furnace running to its full capacity the false wall was torn out and the furnace restored to its original size.

Furnace Charging at the Granby Works.—At the Granby works, Grand Forks, B. C., coke and ore are charged separately into the furnaces.

A train of three charge cars, similar to the design shown in Fig. 215, is taken by a 30-hp. electric locomotive under the bins and receives 1½ tons of coke. This is weighed and charged into the furnace, and the train immediately returns to the bin for the ore charge. Ten tons of ore are run into the cars, weighed and emptied into the furnace. This separate charging of coke and ore, says Frank E. Lathe (*Bull.*, Can. Min. Inst., June, 1910), insures an even distribution of both, and a consequent high fuel efficiency in smelting operations. The cars are supplied with two sets of wheels, as shown in the cut, the lower for ordinary locomotion, and the upper, near the top, for supporting the cars on the heavy rails set in the side walls of the furnace when the cars enter with their load. When the cars are completely inside the furnace, the feeder, by means of a long iron hook, pulls the arm marked *A* in the drawing. This releases the lock, and the contents of the cars fall upon the charge below.

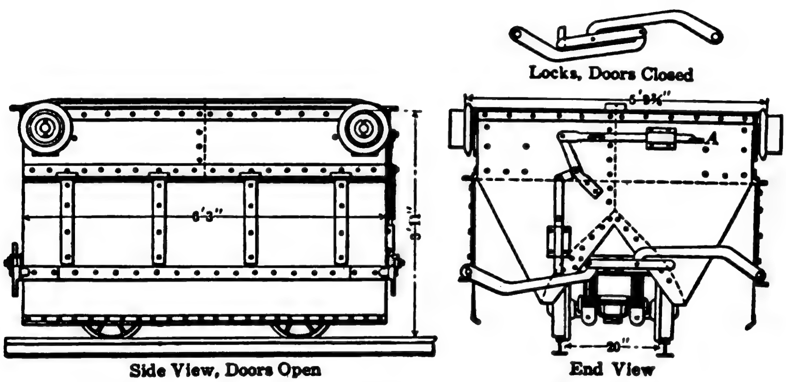


FIG. 215.—FURNACE-CHARGING CAR AT GRANBY SMELTERY.

Usually the locks of two cars are chained together, so that they dump at the same time. In practice, the time occupied from the moment the first car enters the furnace until it is again withdrawn, is from 10 to 20 sec. When the cars are removed, the hoppers are again closed, and the second trip commences. Complete charges of both coke and ore are made every 20 to 30 min. to each furnace, and one motor supplies all the furnaces. This indicates the speed of operation required. As the cars are divided into four compartments, and the motion given the contents is toward the sides of the furnaces, there is no very definite separation of coarse and fine ore. There is a slight tendency, however, for the coarser ore to fall near the center and at the very outside, the fines occupying the two intermediate positions. This is productive of a very uniform descent of charge all over the furnace. Of course, it is easy, in the case of uneven conditions prevailing in the different parts of the furnace, to omit the ore charge from one or more compartments in any car; or to add extra coke as may be necessary.

Furnace-charging Car.—At the Fundicion smeltery of the Pacific Smelting & Mining Co. in southern Sonora, Mex., a device in connection with a furnace-charging car is used, which is of interest and of utility under some conditions. Instead of having a car into which the charge is weighed from the various scales, a side-door, gable-bottom car is used, which is divided vertically into four or five compartments. Each compartment is numbered to correspond with a number on the bins containing respectively the ore, flux, lime and fuel. The metallurgist makes up the charge by calculation and places a mark by means of a peg in a series of holes on the side of each one of the respective compartments, corresponding to the amount of load from each bin which must be taken to make up the calculated charge. The native workman pushes the car, which in this case is on the surface level, under the bins and takes a load up to the respective pegs. The whole car is then moved over a discharge way in the track and is dumped into the elevator boot by opening the side door. It is claimed that this device is fool-proof, an item of some importance in connection with the use of Mexican labor, and also that the arrangement of having the charge delivered in a succession of streams from the side doors results in a desirable and suitable mixture of the charge in the boot of the elevator, from which it is elevated to the charging floor. The device was tried and found to work successfully at the plant above mentioned.

Motor Operated Charging and Slag Cars.—Where larries or charging cars are to be used singly or in trains of not more than two or three, greater flexibility is obtained by having them self-propelled by electric motors. Operations can also be carried on more rapidly because delays occasioned by waiting for the locomotive are obviated. A motor-driven charging car is installed in the lead department of the International Smelting & Refining Co., at Tooele, Utah. This car is equipped with two Westinghouse No. 64 motors, one on each truck, with the controller mounted in a horizontal position at the end of the car. The motors and controller are of the street-railway type somewhat modified for the special service required. The brush holders, commutators and bearings are designed for long service. The lubrication is simple and adequate for long periods of operation. The important characteristics claimed for these cars are ability to develop great power in small space, reliability and ease of inspection. Motor-tilted slag cars are now used at many smelting plants. The slag car used by the International Smelting & Refining Co., has a Westinghouse No. 6, type K, direct-current motor mounted on the end of the car and protected by a canopy. It is series wound, entirely inclosed, and is dust and weatherproof. It is geared to the tilting mechanism and is designed especially to take care of heavy service of this character and withstand the high temperature to which it may be subjected.

A Trapped Charging Bell.—The charging bell shown in Fig. 216 is one used in the electric iron-smelting furnace of Dorsey A. Lyon and Edwin R. Cox at Heroult, Calif., in order to have the furnace trapped at all times. Both the purpose, construction and method of operation are completely shown by the cut. It seems to be one of those metallurgical ideas capable of a wide application.

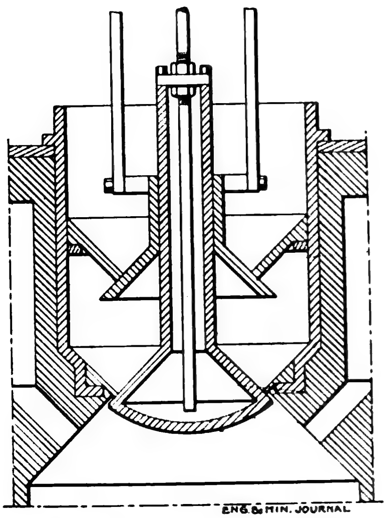


FIG. 216.—DOUBLE CHARGING BELL.

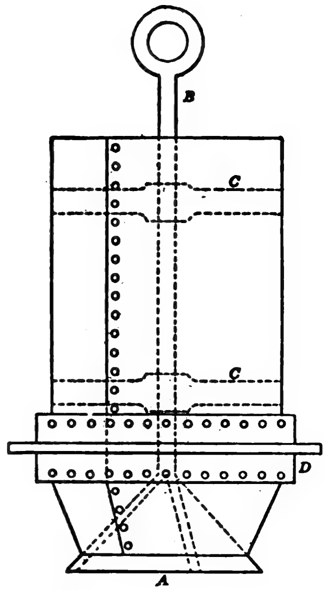


FIG. 217.—KILN-CHARGING DEVICE.

A Kiln-charging Device.—An apparatus used around Chicago iron furnaces for charging dolomite-kilns, seems adaptable to many classes of work. It consists of a cylinder of heavy sheet steel terminating in an inverted truncated cone of the same material. Referring to Fig. 217, cone *A*, supported on the inside by a spider which is fastened to a steel rod *B*, closes the bottom. The rod *B* is free to move in two steel spiders *C*. *D* is a section of T-rail fastened around the body of the charger. In filling the cylinder, the whole device is supported on *A*. When it is to be moved it is picked up by a crane, by means of the ring on top. The tops of the kilns are so arranged that they just fit the T-rail; the device has only to be set down on the furnace and the contents discharge into the kiln. If the cylinder is to be emptied on the ground, blocks are placed to catch the T-rail. By setting it down on a flat-car the apparatus can be transported from one building to another.

Determination of Flue Leakage (By George C. Westby and O. E. Jager).—Investigations in furnace economics often call for the measure-

ment of flue or chimney discharges, and the estimation of the leakage through the masonry of the smoke channels. A calculation for deducing the leakage from the smoke analysis is here given. As it is generally more convenient to calculate the discharge from analysis of the gases and vapors constituting the smoke, this method of deriving the smoke volume is used. As a basis for the computation it is necessary to know the weight of one of the volatilized constituents of the ore or fuel, whose products of combustion form the gases or smoke under consideration. The choice of the constituent would depend on the data available and on the kind of plant to be tested. At a smelting plant the sulphur in a pyritous ore would be the most convenient factor on which to base the estimation, while the carbon in coal might be adopted as the foundation of the computation at a power plant.

Take as an example the case of a furnace roasting pyritous ore, where 11.5 lb. of sulphur are burned off per min. and the flue gas analyzes 4.5% SO_2. Since 11.5 lb. S gives 23 lb. SO_2, and 1 cu. ft. SO_2 weighs 0.1709 lb. at 0° C. and 760 mm. of mercury, the number of cubic feet of SO_2 per min. at 0° C. and 760 mm. would be 134.5, and since 134.5 cu. ft. SO_2 is 4.5% of the total volume of dry smoke, then there are 2985.9 cu. ft. of dry smoke at standard pressure and temperature. Assuming the observed temperature and pressure to be 280° C. and 630 mm. mercury, respectively, then $2985.9 \times \frac{760}{630} \times \frac{553}{273} = 7296.4$ cu. ft. dry smoke at observed conditions. To this volume must be added that of the water vapor arising from the moisture of the ore.

Assuming that the furnace treats 35 tons of ore per day, carrying 5% moisture, there are 2.43 lb. of water driven off per min. into the flue. This is 63.9 cu. ft. at 100° C. and 760 mm., and at the pressure and temperature observed the volume is 113.8 cu. ft. which, added to the volume of dry smoke gives a total discharge of 7410.2 cu. ft. per min., at the observed temperature and pressure.

After the derivation of the volume passing through the smoke conduit at any particular point, the determination of leakage between any points along the flues can be made by running simultaneous analyses for SO_2 at the points between which the measurement of leakage is required. The computation is then as follows:

Let a = number of parts of SO_2 at first point; b = number of parts of SO_2 at second point; and x = volume of leakage. Then, $\frac{a}{1 \times x}$ = number of parts SO_2 per volume at second point, or $\frac{a}{1 + x} = b$; $x = \frac{a - b}{b}$ and $100x$ = percentage of leakage.

To apply this to the assumed case and neglecting consideration of correction for temperature differences, although in general the temperature correction must be made: If there is 4.5% SO_2 at point A, and 3.5% SO_2 at point B, by substituting in the formula we have $x = \frac{4.5 - 3.5}{3.5} = 0.285$, or leakage = 28.5%. If the discharge at point A is 7410.2 cu. ft., 28.5% of this is 2111.9 cu. ft., representing the leakage of air between the two points.

Determining Dust Losses from Roasters (By C. C. Hoke).—In order to determine the dust losses from roasters, lead and calcining furnaces, converters and the like, the Compañia Minera de Peñoles, at Mapimi, Durango, Mex., installed an experimental bag house consisting essentially of a gas-tight settling chamber opening into filtering bags suspended vertically from supports above. The gas to be filtered is drawn from its

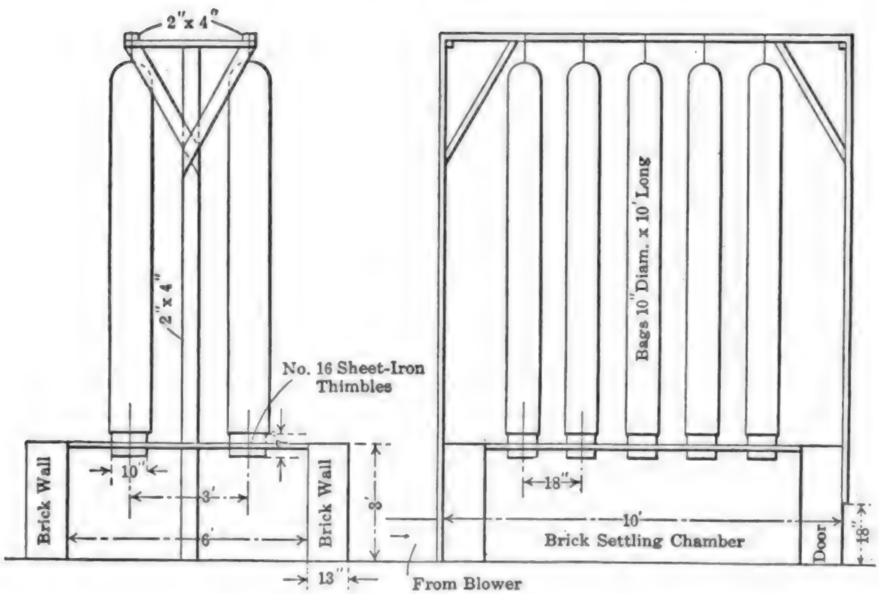


FIG. 218.—EXPERIMENTAL DUST CATCHING PLANT.

source by a fan blower and discharged into the settling chamber, whence it escapes through the bags. Upon cutting off the pressure from the bags and shaking them, the fume falls into the settling chamber and may be removed through a door provided for the purpose. The size of the bag house and number of bags required will depend on general conditions, such as the fume density in the gases to be treated, volume of gas driven through the bags, etc.

In making a quantitative test it is necessary to provide an anemometer

to measure velocities in the fan suction or discharge, and in the flue or source from which the sample is drawn. Care should be taken that the cross-section of the pipe in which the velocity is measured be such that the velocity will not exceed 1000 ft. per min., and a lower velocity (say 600 to 800 ft. per min.) will be desirable. The ratio of the fume recovered by the bags to the total will be as the volume of gas delivered to the bags to that passing through the source from which sample is taken, both volumes being reduced, of course, to an equivalent pressure and temperature. The interval between shaking the bags obviously depends entirely on the fume density in the gases, and should be such that no great reduction in volume delivered by the fan will result. In treating 25 lb. of fume per hour, the interval between shaking the bags would probably be about 1½ hours.

This installation, which can be cheaply constructed and is simple in operation, has proved entirely satisfactory. It should commend itself to anyone desiring to carry on investigations of like nature. The construction of the plant, as illustrated in Fig. 218, is as follows:

The settling chamber is 3 ft. high, 6 ft. wide and 10 ft. long; the walls are built of brick—two bricks thick—and the top or thimble floor is of matched flooring made air-tight by cementing over. The thimbles are 10 in. in diameter, made of sheet iron about 1/16 in. thick, with an expanded rim at the top to fasten the bags more securely, and are fitted tightly into holes cut through the thimble floor. They are further retained by lugs riveted on the sides. There are 10 bags of a fair quality of drilling, running about 50 strands to the inch, and measuring 10 in. in diameter by 10 ft. long. They are secured to the thimbles by tying tightly, and are suspended (not too tightly) from a wooden framework above the thimble floor. A fan blower, suitable for delivering about 1000 cu. ft. of gases per min. at a pressure of ½ in. of water is used, delivering to the fume chamber through a 9-in. pipe. A cleaning door, 18 × 24 in., is provided at one end of the chamber. This apparatus is capable of handling about 25 lb. of fume per hour, if the bags are shaken at frequent intervals.

Dust Determination by Filtration through Sugar.—In testing air in the mines of South Africa, A. McArthur Johnston tried various filtering media, and after comparison it was decided that better results could be obtained with cane sugar than with cotton wool, glass wool, or water (*Journ.* Chem., Met. and Min. Soc. of So. Afr., May, 1912). The filter bed is obtained by coarsely crushing lump sugar, and using that which fails to pass a 90-mesh (0.006-in. aperture) sieve. The filtering layer is made about 2 in. thick and may require a copper or brass gauze to hold it in place. It should be slightly moistened, to aid in the retention of dust, but not sufficiently to cause caking. This point has to be guessed at. To

estimate the dust, after the air is aspirated through the sugar, the sugar is dissolved in distilled water, and then the solution filtered through a tared filter paper or on asbestos in a gooch. The weight found represents the total dust present. Incineration will then give a difference, organic matter.

Water Spray for Dust Settling.—According to the report of the British Alkali Inspector for 1912, much improvement has resulted at smelting works from the use of water sprays to assist in the deposition of the finely divided solid particles carried away from the furnaces along with the smoke, but the diminution of the amount of acid gases sent into the air from many works of this class is regarded as a problem still awaiting a satisfactory solution. Experience suggests that it would be advantageous to bring works in which blende is calcined under the same regulations as now apply to arsenic works treating ores containing a considerable proportion of sulphur. Objections have been raised to the use of water spray in large flues or chambers for the removal of fume, etc., on the ground of the reduction of temperature involved, but it is pointed out, as not generally known, that the weight of gas or air removed by a chimney is almost at its maximum at a temperature of 250° F. (175° C.) above the atmospheric temperature, and observation of the temperature of the chimney gases is recommended as an easy way of controlling the amount of water spray to be used. Water-spray jets should be in such a position that the spray is at once directed into the full current of the gases and carried along with them.

Dust Chamber Velocities.—Discussing flue-dust losses at a meeting of the Institution of Mining and Metallurgy, Lewis T. Wright said that from the top of the short stacks used 20 years ago there might have been a loss of 10% in flue dust. Many years ago he had made some experiments, and found that an air velocity of 16 ft. per sec. would freely carry coarse flue dust from rest, and a velocity of a little less than 10 ft. per sec. would hardly move it at all. He thought at that time a velocity of 5 ft. per sec. was a tolerably safe velocity in the dust-collecting flues. He commented on the fact that Doctor Douglas, in his paper on the Copper Queen mines, had spoken of a velocity of 150 ft. per min. (2½ ft. per sec.), which had been found at Douglas to be sufficiently low to settle very fine flue dust in comparatively short distances. He inferred that in the new plant Doctor Douglas' company had adopted a velocity of 5 ft. per sec. because reference was made by Doctor Douglas to the wire screens giving equally good results at twice the previously mentioned velocity. It was considered that at this velocity practically all the dust discharged from the furnaces would settle out if given an opportunity.

Mr. Wright recalled the school experiments of the large glass globe with two small inlets. Through that glass globe was blown smoke which,

instead of diffusing itself throughout the globe, took a straight course from the inlet to the outlet on the other side; that was to say, the gas or air had inertia. Therefore, in introducing air into the large chamber for the purpose of diminishing its velocity, it had to be given an opportunity of extending itself through the whole section of the chamber, and he did not think that was always done. Such precaution, however, had been taken in the case which he cited where a velocity of 5 ft. per sec. had proved practicable. In some cases where the gases carried away metals more valuable than copper, he thought that a velocity of 1 ft. per sec. would be the least that could be adopted.

Slag Handling Arrangement at British Columbia Copper Co.—To handle economically and efficiently the flow of slag during the interval of changing pots, the British Columbia Copper Co., at its Greenwood plant uses an auxiliary slag bowl thus eliminating the necessity of a second spout on the settler. This simplifies the track layout for slag disposal and avoids the spilling of slag.

The slag-handling arrangement at this plant consists of two parallel tracks on which an electric locomotive handles the 225-cu. ft. motor-dumped slag pots. Short spur tracks connect the two parallel tracks at each settler and the locomotive, returning from the dump with the string of empties, takes the full pot from the settler to the outside parallel track, "kicking" it a short distance down the track, which is level at this point but has an upgrade as it approaches the dump proper. The locomotive then leaves an empty pot at this furnace and passes on to repeat the operation at the next.

During the interval of changing the large slag pots, the auxiliary slag bowl is swung under the settler slag spout and receives the slag flow until the new pot is in position. The auxiliary bowl is usually only filled to about one-third its capacity so that there is ample allowance for emergencies and delay in changing the pots.

The device used to facilitate the handling of the slag is simple and merely consists of a bowl of ample capacity to be swung under the slag pot during the period of changing pots. It is an adaptation and amplification of the old hand-ladle idea with sufficient capacity provided and suitable means for sustaining and moving the larger bowl. The auxiliary bowl is approximately 2 ft. deep and is elliptical in shape with diameters of about 4 and 5 ft. The bowl rests in a cast-steel frame and is held in place by lugs. The cast-steel frame is socketed in the swinging cast-iron frame. This cast-iron frame has ball bearings at both top and bottom, the base resting on a cast-iron seat in the floor and the top being fastened to a girder. The frame and bowl operate easily on these bearings and could be readily moved by a boy. The center of gravity of the auxiliary bowl is so placed that it tends to right itself when empty, and to dump

when full. When receiving slag, it is held in position by a pawl in the ratchet near the handle.

This slag-handling arrangement has proved highly efficient at this plant and it is said that the blast has not been taken off the furnaces in two years on account of the blocking of the slag flow. The auxiliary-bowl arrangement was installed at the British Columbia Copper Co.'s smeltery in the autumn of 1906 and is not patented. We understand that a similar equipment is now used at the plant of the Canadian Copper Co. at Coppercliff, Ontario.

An important consideration in the use of this auxiliary bowl is the fact that at no time does the slag stream impinge against the metal of the large and expensive slag pots. When a new slag pot is placed at a settler, the slag caught in the auxiliary bowl during the interval of changing pots is turned into the new pot at once, forming a pool or buffer to receive the slag stream of the settler. The bowls of the slag pots at the British Columbia works have not been replaced in two years and the incidental saving in this connection is noteworthy.

Cleaning Blast-furnace Slag.—At the works of the Mammoth Copper Mining Co., at Kennett, Shasta county, Calif., some experimentation has lately been carried on for the purpose of determining a method of cleaning the blast-furnace slag. A 13 × 30-ft. reverberatory furnace has been built and tried as an extra settler. This small reverberatory is fired by oil burners. Its capacity is about 300 tons of slag per day. At first, from 7 to 8 tons of sulphide fines were added each day to the charge in this settler. The desired clearing effect upon the slag was not obtained through the use of the sulphide, so that this practice was discontinued and lately the reverberatory has been used merely as an extra settler. No exact data are available as to the results obtained, but it is understood that the extra settler has accomplished what was expected, namely, to determine just what grade of slag it would pay to treat in an auxiliary reverberatory furnace.

Copper Blast-furnace Settlers.—Large settlers are now commonly used for receiving molten matte and slag from modern copper-blast furnaces. They are either round or oval in shape, the round ones often being 16 to 18 ft. in diameter and 5 ft. deep, while some of the large oval settlers are 22 ft. long, 14 ft. wide and 4 ft. deep. A better separation of matte and slag is claimed for the oval settler as the slag enters near one end and discharges at the other after traveling a distance of about 18 or 20 ft. These large settlers are lined with chrome brick with a backing of ganister or converter lining. Chrome brick is generally preferred to either fireclay or magnesia brick on account of its durability. It is practically unaffected by any of the slags or mattes met with in copper smelting although it is possibly not quite so effect-

ive in withstanding the chemical action of some mattes and slags as magnesia brick. But chrome brick will withstand variations of temperature without scaling off as magnesia brick is liable to do. A chrome brick which has given much satisfaction in copper work has an approximate analysis as follows: Silica, 3%; alumina, 24%; ferric oxide, 16%; magnesia, 14%; chromic oxide, 40%. In one of the best designed copper plants consisting of three large blast-furnaces, oval settlers are used 18 ft. long, 10 ft. 6 in. wide and 4 ft. 6 in. deep. They are placed between the furnaces and at each end of the line, making four in all. This arrangement permits of repairs to any furnace or settler without interfering with the work. The settlers are lined with chrome brick and are cooled externally when necessary by sprays of water running down the steel sides. The slags are noticeably cleaner than before, when smaller settlers were used.

Breaking Up Slag in Reverberatory Furnaces.—Sometimes in reverberatory-furnace work a charge too high in silica gets into the furnace, and before basic ore can be added to flux it and make the slag fluid, a silicious crust has formed over the top of the charge. This makes it hard for the heat to get through to the charge underneath, and trouble soon ensues. At the reverberatory furnaces of the International smeltery at Tooele, Utah, provision is made for the quick removal of all such troubles. These furnaces are built like those at Anaconda with an air space in the bridge of the furnace for a current of cool air to circulate so as to cool the conker plates. But, lest this should not be enough, an air jet is provided for blowing in compressed air to cool them. This air pipe is now extended along the side of the furnace so that a hose can be connected at different points. Whenever a crust forms over the charge an iron pipe is connected by a hose to the compressed-air pipe and poked down through the charge. The air, which is at a pressure of 90 lb. per sq. in. is turned on, and some of the iron in the matte is oxidized and rises to the surface to flux off the siliceous crust, while the violent agitation breaks up the crust itself. In this way the furnace is soon brought back to a normal condition.

Iron and Steel Mending (By Claude T. Rice).—Recently the Mountain Copper Co. had the Mammoth company patch the broken frame of a locomotive by means of thermit, and although subjected to considerable strain the patched place seems to be almost as strong as the rest of the frame. Holes are burned through ladles by matte, or are worn through other apparatus about the smeltery; when the rest of the apparatus is still in good condition, the hole is easily patched by means of thermit, and the repaired article may be used much longer. Holes 6 in. in diameter in ladles have been mended by this method. Pan conveyors are used about the Mammoth plant for conveying and casting

matte and converter slag. Small holes are burned through the bottom while the rest of the pan is still in fair condition. These holes are easily mended by means of the oxy-acetylene blowpipe and soft Swedish iron for "solder." The blowpipe is also used for other jobs, such as welding tuyères to jackets and other such repairs. The results with the blowpipe are more easily and cheaply obtained than with the thermit process so in case the mending job is not too large for the blowpipe, that apparatus should be used. With the thermit process, unless the man doing the job is familiar with the work, a leak is liable to occur. The oxy-acetylene mending requires considerable acquaintance with the apparatus and at first the man using it will have poor success, but as he becomes acquainted with the work, better results are obtained. Where not more than 2 cu. in. of the metal are required to make the patch it will in general be found cheaper to use the blowpipe method than the thermit process.

Straightening Furnace Jackets (By Claude T. Rice).—In time the water jackets on blast-furnaces get warped out of shape and either have to be replaced or straightened. Both at Chrome, N. J., and at Kennett, Calif., special furnaces are used for expediting this straightening of the jackets. At the Chrome plant the jacket is slid into the furnace, while at the Mammoth, the jacket is lifted into and out of the furnace by means of a track derrick. The latter furnace is fired by means of oil as that is the cheapest fuel available at the Kennett plant. The top of the furnace is made in sections, the bricks being held between angle irons forming a flat arch. The table on which the jacket is placed for straightening consists of two heavy cast-iron plates, each a little wider than the jacket. These rest on three I-beams which run longitudinally under them, while these beams in turn rest on shorter cross I-beams. From the ends of the cross I-beams extend round rods threaded for a considerable distance at their upper ends. On the upper ends of these rods nuts with long ears are screwed. The red-hot jacket to be straightened is laid on the frame. Then I-beams are laid across the top so that as the bolts are tightened the beams are pressed down and the jacket is straightened. Long-handled wrenches are used in tightening the nuts. The straightening of the warped jackets starts some of the joints, and the jacket always requires some reriveting to stop these leaks. In order to test for leaks the jacket is connected with some supply of compressed air, such as the pipe taking the blast to the converters. The cost of straightening a jacket, of course, depends entirely upon its condition but a badly warped jacket usually can be straightened and reriveted for between $50 and $60, so it has been found to pay well to straighten jackets at the Mammoth plant where machinists get $3.87 per 9-hr. shift.

Withdrawing Stuck Bars.—It frequently happens that when the fore-hearth is full of matte, and requires tapping in a hurry, a bar will have the

head knocked off, says A. A. Summerhayes (*Min. and Eng. Rev.*, Australia, June 5, 1913). Fig. 219 shows a useful tool for drawing the bars, and the following description should make the construction and operation of the apparatus clear. The tool consists of a steel bar, generally made out of 1-in. hexagon steel, with a collar welded solidly on the end and bored about $\frac{1}{8}$ in. larger than the bars in use. A slot is then cut $1\frac{1}{2}$ in. long, $\frac{3}{8}$ in. wide, and a steel wedge fitted in it flush with the outside of the collar so that a space is left in the inside of the slot of $\frac{3}{8}$ in. on the front end of the slot and $\frac{5}{8}$ in. on the back end. In this slot is dropped an ordinary piece of $\frac{1}{2}$-in. round file or steel roller $\frac{5}{16}$ in. long, the ends slightly conical, to prevent the roller dropping through into the bore on the tool. After

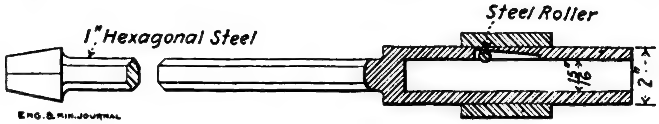


FIG. 219.—APPARATUS TO DRAW STUCK BARS.

putting in the roller, a steel collar is shrunk over the slot. The operation of drawing the bar in the event of the head having broken is as follows: The tool is slipped over the bar, the roller riding on the bar until checked by coming to the end of the bore. The tool is then pulled in the direction it is desired to draw the bar. The roller immediately rides up the wedge and so grips the bar. The harder the bar is driven the tighter it naturally grips. This machine has pulled a $\frac{7}{8}$-in. steel bar in halves, the tool gripping tight all the time. The advantage of the tool is that a tap with a hammer in the opposite direction will immediately release the implement, which is free to be used again.

Refractory Furnace Lining.—For certain purposes a furnace lining of a refractory nature may be made from asbestos and water-glass, says the *Brass World.* It is useful for patching or plugging cracks as it does not crumble as readily as other similar compositions made from clay. The material is made of the following: Fine asbestos, 40 lb.; water-glass, 60 lb. The water-glass is the same as silicate of soda and occurs in commerce as a jelly-like mass which is soluble in water. The asbestos and water-glass are mixed with enough water to make the whole pasty so that it can be worked.

Strength of Adobe Brick.—It is reported by *Engineering News* that tests on the strength of adobe brick were made some years ago by Herbert N. Alleman (now a resident of South America), assisted by W. F. Schaphorst, the latter of New York City, but at that time resident in the Rio Grande Valley. As is well known, adobe is the alluvial clay of southwestern United States and Mexico, which is formed into brick (usually 9 × 12 × 4 in. in size) and allowed to be sun dried. Mexican brickmakers

generally add chaff or some other fiber substance for binding together the mixture. The ingredients are mixed without the aid of machinery. Peons tread in the wet mixture barefooted until the proper consistency for molding has been reached. When dry, adobe has the appearance of an ordinary chunk of dry, gray mud, but it will give a metallic ring when struck, similar to the ring of ordinary burned-clay brick.

The tests were made in tension on briquettes of 1-in. section, of the standard cement-testing type; in compression on 1-in. square by 5-in. long prisms (in compression along the 5 × 1-in. surface); and in flexure on 1-in. square pieces over a span of about 4 in. The average tensile value of the briquettes was found to be about 60 lb. per sq. in., few falling below that figure and some attaining a tension as high as 80 lb. per sq. in. The compression specimens ran up to about 500 lb. per sq. in. on the average, with not very great variation. The flexure specimens broke uniformly in tension with an extreme-fiber stress value of about 60 lb. per sq. in. All specimens were carefully made and were undoubtedly of considerably greater strength than would be attained by the materials commonly used and mixed in the usual way.

Adobe brick is an excellent building material for many purposes, as engineers who have operated in Mexico well know. There are adobe houses of great age still standing in Mexico, and showing but little deterioration. Much depends, of course, upon the character of the adobe. Ill-made brick of sandy mud may not last through a single rainy season, while properly made brick of a good kind of mud may stand exposure to the weather for 100 years or more. The authors of the statements quoted by *Engineering News* evidently referred to good brick. The high strength against compression that they report, viz., 500 lb. per sq. in. on the average, is surprising. This reminds us of an experience related by a smelter of our acquaintance. He built a battery of roasting stalls out of adobe brick in default of any other material. However, he found adobe brick to answer so well for this purpose that later on he would not have chosen any other material even if he could have had it. At one end of the line of roasting stalls, he turned up a little nub of a chimney. Of course, the draft was poor. In order to improve it, he raised the chimney 10 ft. at a time, being fearful to do very much lest the whole thing might crumble and fall. Having attained a height of 40 ft., his courage gave out and he stopped there. Now, a chimney of 40 ft. in height would have a weight of only about 43 lb. per sq. in. Assuming Mr. Alleman's figure of 500 lb., there was, of course, a large factor of safety. We should not, however, advise too much reliance upon any single figure of this kind, knowing very well the wide range of difference among several kinds of adobe brick. The dimensions of the Mexican adobe are frequently 4 × 8 × 16 inches.

A Simple Charcoal Oven (By A. Livingstone Oke).—The accompanying notes show a type of charcoal oven commonly used in Chile and elsewhere in South America. The oven is excavated in a bank of gravel sufficiently cemented to stand without flaking or scaling. Ordinary recent alluvial or glacial drift is suitable for the purpose. The oven is charged by first laying small brushwood in the air channel shown along the bottom in Fig. 220 and the floor is also covered over a few inches deep with the same material. The sticks of wood are then laid across the oven, on top of the layer of brushwood. When the oven is full almost to the mouth, allowing space for the double wall of turf or stone, a quantity of dry twigs and easily burning wood is put in the front and ignited. This fire in the mouth is kept going for an hour or more, fresh fuel being added to prevent too much of the wood inside from being consumed. This preliminary firing is to get the charge thoroughly heated, and during this stage the greater bulk of the gases and volatile matter come off in dense clouds of dark smoke. When considered to be properly ignited, the

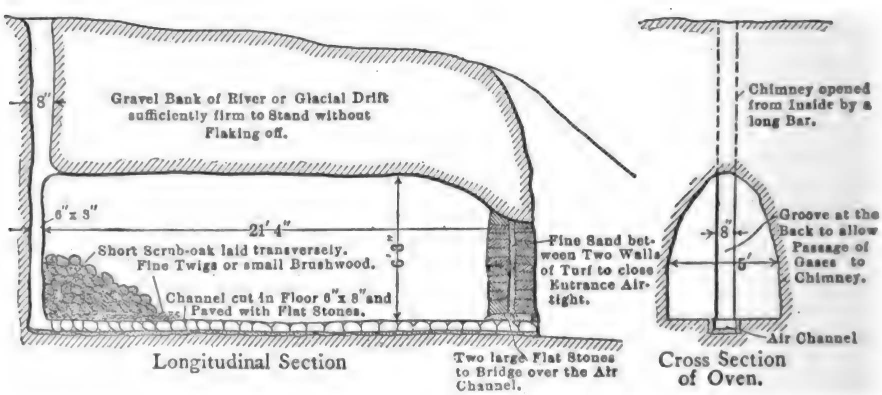


FIG. 220.—TYPE OF CHARCOAL OVEN USED IN THE ANDES.

double wall is built, the inner part being first completed up to the roof and then the outer wall built, fine sand, clay or earth, being rammed tightly between them, to render the mouth air-tight, except the small air channel, left for the admission and control of the air along the bottom of the oven. The operation of closing the mouth is the more important because on it depends the success of the whole firing. If insufficiently stopped, air will enter, resulting in the reduction of wood to cinder and loss of most or all of the charge. The air channel at the bottom is left open until the smoke coming out is a thin white color, which may be in from 12 to 24 hr., and then this opening is also carefully sealed up. The upper end of the chimney is left open a little longer and finally closed with clay. The oven is now left three or more days, to cool; if it is opened too soon

there is a risk of the charcoal starting to burn again. To obtain this maximum output of charcoal, I found it desirable to modify the oven and the mode of operating it as follows: A second chimney was put in the center of the oven. The front of the furnace is now carefully closed, air-tight, and the charge fired at the bottom by means of the entrance to the air channel left for the purpose. On first firing, the chimney at the back is kept closed until only white smoke issues from the one in the center. Then the latter is closed and the back opened. Finally all three openings are closed and the oven left to cool. In the matter of charging the wood, it was also found better to stand the wood upright with the bigger ends uppermost. These various modifications on the usual native method of operating increased the output from 16.4% to 23.7 per cent.

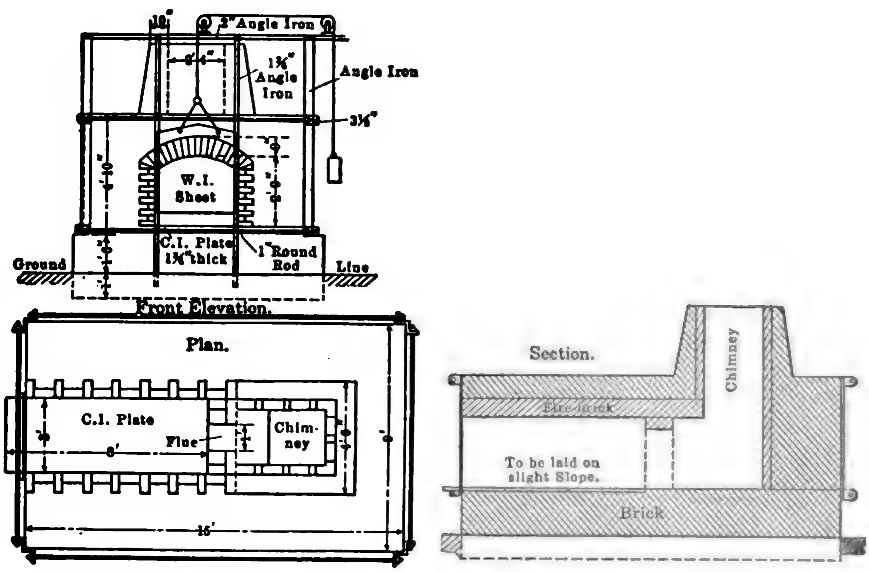


FIG. 221.—OVEN FOR MAKING SOFT COKE.

A Small Coke Oven.—In the report for 1909 of the chief inspector of mines in India there is described a small oven for making soft coke, the details of which are shown in Fig. 221. The 1¾-in. cast-iron sole-plate should be made in two pieces, each 4 × 3 ft. The object of the plate is to retain the heat after the charge has been started, so as to raise the unburnt coal quickly, as it is turned over in the oven, to a comparatively high temperature. The rest of the sketch is self-explanatory. Coal of about 1½-in. cubes is best suited to the production of soft coke by this oven, but it should be remembered that only coals which will fuse can be used successfully. Carbonization is facilitated by gently lifting the charge every 15 min. or so to admit air into the interior of the charge by means of a

pricker or pointed long iron bar, and should be complete in from 6 to 8 hr., according to the volume of volatile matter to be driven off. If the oven is to be used for the production of hard coke, it should not be forgotten that the last traces of volatile matter must be expelled; if soft coke is to be produced the charge must be withdrawn before the last traces are driven off. Out of 100 tons of coal it is possible to get from 65 to 70 tons of soft coke by the use of this oven.

COPPER SMELTING

Jacket Water for Copper Blast-furnaces.—According to Percy E Barbour, two blast-furnaces, one 72 × 180 in., and one 56 × 180 in., smelted during one 24-hr. period the following: Green ore, 6 tons; roasted ore, 492; converter slag, 102; blast-furnace slag, 32; silicious flux, 75; matte, 24.25; total, 731.25 tons. Total coke used, 83.04 tons. During this period the jacket water was carefully measured and averaged for the two furnaces together 565 gal. per min. The temperature of the water fed to the jackets was 74.5 deg. F., and the temperature of the jacket water in the discharge launder was 110 degrees.

Cast-iron Tuyeres (By Bancroft Gore).—At the copper smeltery of the Compañia Minera de Gatico, Chile, the customary thimbles beaded over the inner wall of the jacket proved troublesome owing to the seepage of jacket water between them and the jacket plates. The furnace had to be blown out on several occasions to expand these thimbles and to calk the leaks between them and the jacket walls. Whenever a leak occurred on the fire side, unless the furnace was blown out at once and the beading calked, the corrosive action of the sea water used in the jackets was such that the plate in the region of the seepage became thin, making further repairs impossible and a change of jacket imperative. These leaks generally appeared 3 or 4 four months after the installation of a new jacket, and in spite of repeated calking, with incidental interruption to all smelting operations, the life of a jacket was short owing to defective tuyere construction.

As facilities were not on hand for welding the thimbles to the jacket walls, cast-iron rings riveted to the jacket plates were adopted in spite of the usually accepted theory that the fire wall of a blast-furnace is no place for rivet heads, and that the rapid driving of the furnace, and the corrosive nature of a free-flowing slag would lead to worse difficulties at the tuyeres than had been experienced from the use of the discarded thimbles. The furnace smelts from 4.6 to 5 metric tons per sq. ft. of hearth area per 24 hr., and the cast-iron tuyeres have proved superior under the conditions present at this plant.

The rivet holes on both jacket plates were countersunk and special care

taken on fire side to avoid rivet heads projecting above the surface of the plate. Each jacket was tested to 30 lb. hydraulic pressure, and in February, 1909, the furnace, equipped with these jackets, was blown in. During nearly 2 years of service there has been no seepage at the tuyeres from either wall of the jackets, and after an inspection made recently during the New Year's feast days, when the bed of the furnace was removed, the rivet heads on the fire side of the jackets were found to be in as good condition as when first installed, and what was formerly the weakest point of the jacket was judged to be the most permanent and indestructible part of the furnace. The massive nature of the rings is especially advantageous where furnace men are inclined to be careless in the use of bar and sledge at the tuyeres.

Relief Valves for Blast Pipes (By Percy E. Barbour).—The relief valves shown in Fig. 222 are used at two copper smelteries in this country.

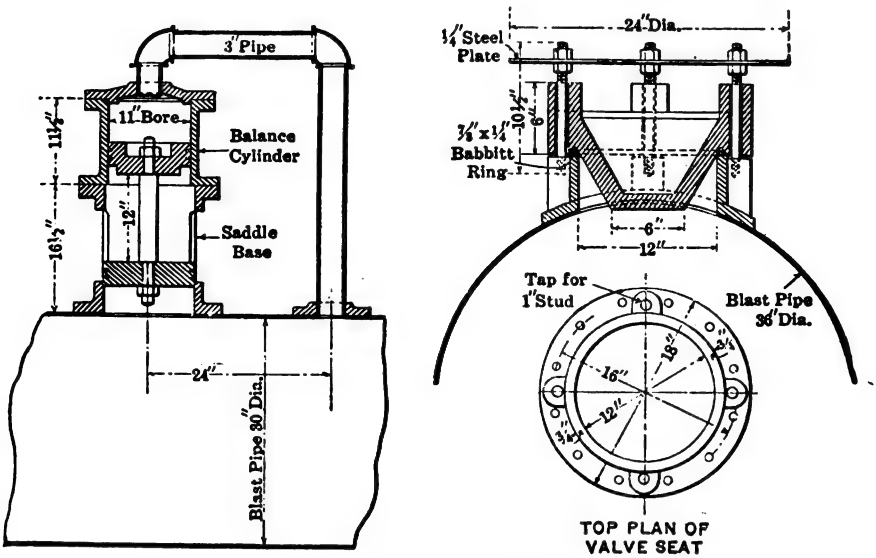


FIG. 222.—TWO TYPES OF BLAST-PIPE RELIEF VALVES.

The balanced-valve type, shown on the left, is being used on a converter blast pipe with an air pressure of 15 lb. per sq. in. The valve can be adapted to higher pressures by loading the lower piston. This valve works admirably, floating up and down freely with the fluctuations of pressure in the blast pipe. The poppet-valve type, shown in the right-hand drawing, is being used on a blast-furnace blast pipe with air at 30 oz. per sq. in. This valve can also be used for higher pressures by loading the steel-plate shield at the top.

Both these valves have stood the test of long usage and answer well

the purpose of relieving sudden and excessive pressure in the blast pipes while the engine, either by automatic regulator or by hand throttling, is being slowed down.

Conker Plate Details (By Percy E. Barbour).—Various types of conker plates for the bridge walls of reverberatory furnaces have been tried with the varying results to be expected. The Garfield type of conker plate, which is now giving perfect satisfaction, is the result of consistent and continual development since the first reverberatory was built at this Utah plant in 1905. The first conker plates used in furnaces Nos. 1 and 2, and shown in Fig. 1 (referring to Fig. 223), were built up of four 24-in., 80-lb. standard I-beams laid horizontally and fastened rigidly to two ½-in. plates standing vertically against the flanges of the beams. This conker plate failed by crippling, or better said, by warping of the webs, as it was the excessive heat rather than the pressure of the furnace burden which caused the failure. The latter was of course only gradual and did not cause a catastrophe. The total cross-section is so

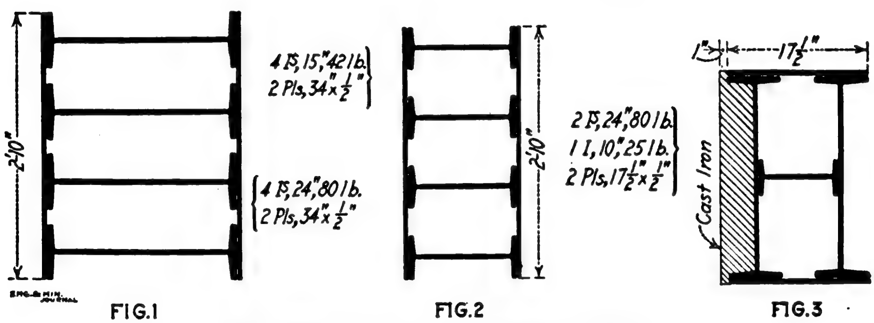


FIG. 223.—DETAILS OF CONKER PLATES.

great that unless brick-work which would make the bridge wall of prohibitive size is used, the steel section does not get sufficient protection from the heat.

On furnaces Nos. 3 and 4, the plate shown in Fig. 2 was used. This consisted of four 15-in., 42-lb., standard I-beams with two ½-in. plates built up as the former plate was. This gave a greater thickness of brickwork against the sides but no more on the top and the plate was not wholly satisfactory, due again to crippling of the webs of the beams. Furnace No. 5 had a conker plate made of three 10-in. 25-lb. standard I-beams arranged as shown in Fig. 3, with a cast-iron plate on the furnace side, and two ½-in. plates on the flanges, at top and bottom. This section gave better results, but the air spaces were too small for the circulation required and the I-beams were changed for the No. 6 and last furnace. This conker plate for the No. 6 furnace is excellent because it does the work and it does the work because it is excellent mechanically. It is

composed of two 24-in. 80-lb. standard I-beams, shown in Fig. 3, standing vertically, with a 10-in. 25-lb. I-beam laid horizontally along the middle of the webs of the others. This small beam stiffens the long webs of the larger beams and prevents warping. The top and bottom of the steel section is completed by two ½-in. plates. All these members are riveted rigidly together. The furnace side of this plate has bolted to it a cast-iron plate thick enough to extend 1 in. beyond the flange of the 24-in. I-beam, as shown. This cast-iron plate is made in sections with the joints vertical and halved into each other so that at the joints the I-beams will not be unprotected. The blast pipe for the blast-furnaces is tapped and a small pipe delivers air at one end of this conker plate which blows through the other end and keeps up a circulation. This conker plate is satisfactory.

Trap Spout for Copper Blast-furnace (By Arturo Poupin).—Trap spouts are not new, but their design involves some interesting details. The sketch to the left in Fig. 224 shows our old design that must have met with difficulties wherever used, and probably never worked satisfactorily, and the reason can be explained by the impossibility of keeping a stream

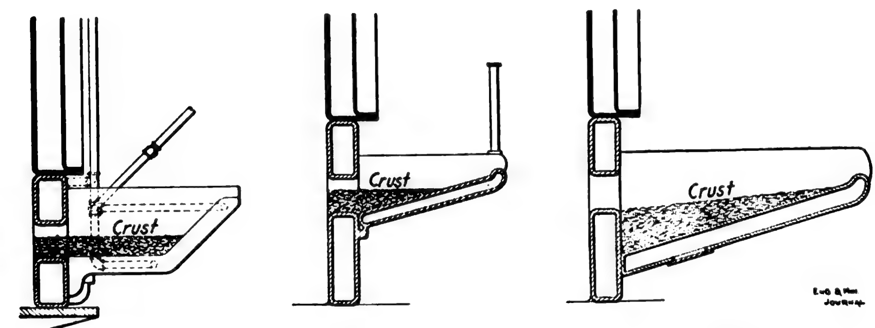


FIG. 1 FIG. 2 FIG. 3

FIG. 224.—EVOLUTION OF THE TRAP SPOUT.

of hot liquid in contact with the cooled surface of the spout, causing the formation of a crust that would stop up the tap hole of the breast jacket. After that, not having an outlet, slag and matte would soon fill up to the level of the tuyeres and stop the working of the furnace. To obviate this fault the later design provides space below the tap hole that can be kept filled with brasque, preventing the formation of a crust. Another point also worth remembering is that the opening in the furnace wall is made ample in the last design, so that any iron sow can be taken out easily. Another spout of still later design is made of ⅜-in. steel plate, with the addition of a copper nose at the end. Two ⅜-in. plates are bent to shape and riveted outside all round the sides, and an inlet of 1½ in. at the bottom and

two outlets for the water at the top are provided; on the lower part there is a cover bolted to the shell which serves for cleaning the salts and sediment that is carried by the sea water. The copper nose is composed of a 1-in. water pipe imbedded in copper; water enters one side and leaves on the other. The sketches in Fig. 224 illustrate, from left to right, the evolution of the trap spout at Gatico, Chile.

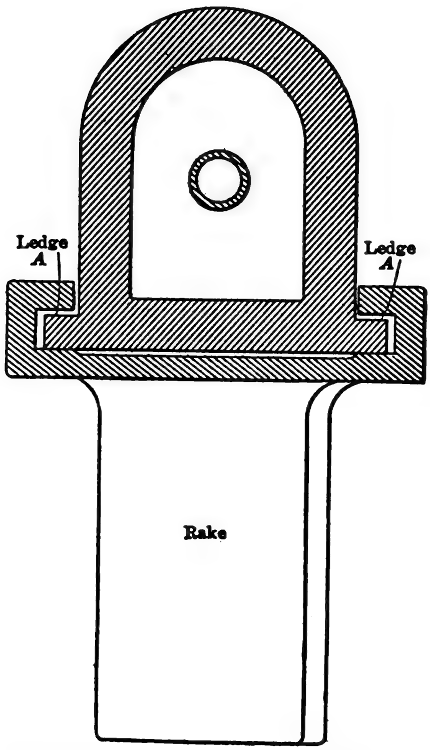


FIG. 225.—ORDINARY MCDOUGAL RABBLE ARM AND RAKE.

Improved Rake and Arm for McDougal Furnace.—In roasting ores with the McDougal furnace, one of the constant sources of expense is the replacement of the rake arms. The standard rake (shown in Fig. 225) becomes sulphated up to the ledge *A*, causing it to break, and frequently it is also necessary to break and remove all the outer rakes if an inner one is to be replaced. To overcome this, an arm and rake, covered by U. S. patent 940,488 (Fig. 226) were designed by Capt. William Kelly, general foreman, and H. N. Thomson, former assistant superintendent, of the Washoe smeltery. The illustration shows the method of attaching the rake, and the water-cooled dovetailed surface of the arm that acts as the support for the rake. This combination has been in use for nearly two

years, and has been satisfactory, as there is no adhesion, of any moment, between the rake and the arm.

Matte Explosions (By W. C. Smith).—The cause of explosions of molten matte when brought in contact with wet or damp surfaces has been an interesting subject to me for several years. Many say that the sudden generation of steam plays an important part; others claim that a rapid partial decomposition of matte due to superheated steam liberates large volumes of gas and this causes the explosion; and still others claim that both steam and gases are responsible. The following are some of my observations: (a) In blowing in a lead blast-furnace we use "mill cinder" (puddling-furnace slag), assaying 40 to 45% Fe and 30 to 33% SiO_2; CaO, none; Al_2O_3, none. When the furnace starts slagging the mill cinder comes through practically unchanged. This molten mill cinder has a marked tendency to explode on contact with damp surfaces. (b) Molten salt cake (Na_2SO_4) has to be handled with extreme care. A damp tool or slag pot will cause a violent explosion. (c) A copper-sodium

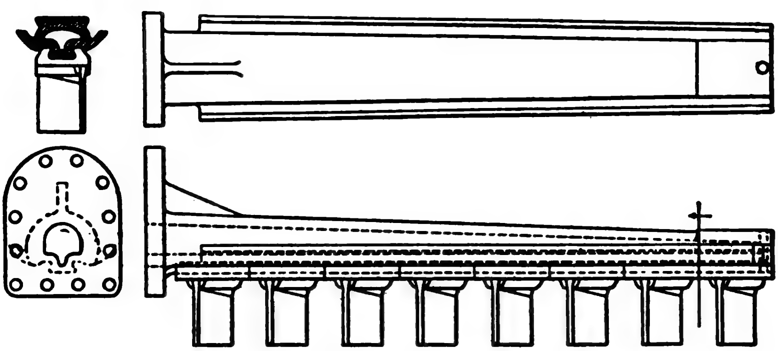


FIG. 226.—NEW MCDOUGAL RABBLE ARM AND RAKE.

matte carrying no iron is more dangerous to handle than ordinary mattes. (d) Molten caustic soda (NaOH), at high temperatures exhibits the same property. (e) Molten soda ash (Na_2CO_3), at high temperatures also requires care in handling. (f) Antimony litharge, consisting of PbO, Sb_2O_3, As_2O_3, and low in SiO_2, explodes violently, if it be at an orange heat.

My conclusions from these cases are: (1) The tendency of these materials to explode is not due to the decompostion of sulphides by steam since five of them contain no sulphides. (2) The presence of compounds of iron is not necessary, as five of them carry no iron. (3) Sodium or its compounds are not essential; for ordinary blast-furnace matte, mill cinder and antimony litharge are free from sodium. (4) No one element or compound is present in all materials under consideration, hence

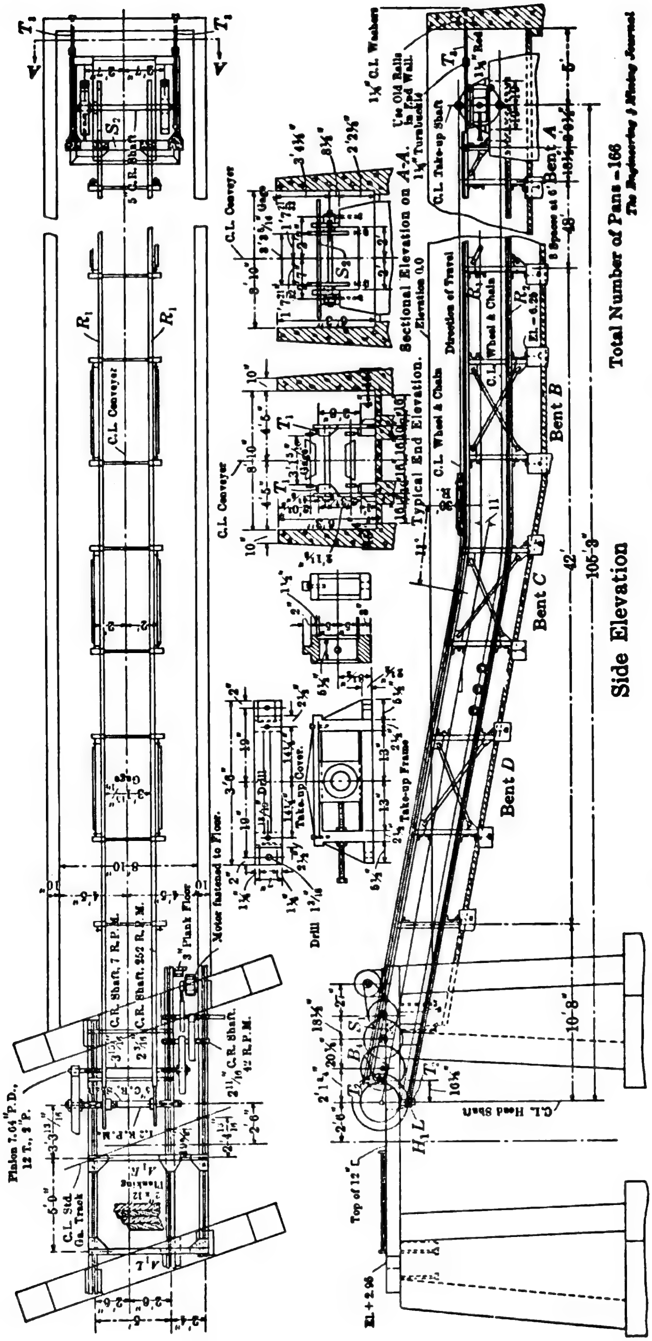


FIG. 227.—DETAILS OF CONVEYOR FROM MATTE-CASTING MACHINE.

explosions are not due to any one element or compound. (5) One property possessed by all these materials when conditions are correct is great fluidity. Great fluidity means low viscosity and low cohesion. (6) A small quantity of matte from a settler was poured on a wet iron plate. The matte exploded violently, flying in all directions. A small quantity of slag from the settler under same conditions did not explode but became pasty, blisters or bubbles rose to the surface, burst, emitted a jet of steam and the blister closed. The low cohesion of the matte allowed the steam to exert its force in scattering it, while in the case of slag the entire force of the steam was absorbed in overcoming the cohesion of the slag sufficiently to permit the steam to escape from blisters. Therefore, I believe that little or no decomposition takes place, but that the explosion is due to the instantaneous generation of steam and the low cohesion of matte.

Matte Conveyor at Mammoth Smeltery.—The details of the take-up frame and the method of driving the casting conveyor which is used in casting the low-grade matte for resmelting at the Mammoth plant of the United States Smelting, Refining & Mining Co., at Kennett, Calif., are shown in Fig. 227.

The method of anchoring the take-up device in the pit is to be noted, as considerable trouble was experienced at first in securing the frame in the pit. The driving gear is driven on to the main shaft and shearing pins are used in it, so as to save the device from serious injury in case something catches or breaks. The utility of these shearing pins has been demonstrated during the operation of the machine, they are readily replaceable and the structure carrying the gear wheels is substantially made to take the strain when any obstacle prevents the moving of the conveyor, resulting in the breaking of the pins.

Matte Conveyor Pan.—At the Mammoth plant of the United States Smelting, Refining & Mining Co., where the ore is smelted semi-pyritically, it is necessary when all the furnaces are being forced to resmelt the first matte on account of the large amount of iron coming down. The matte is tapped into a ladle and then poured into the matte conveyor. The details of the design of the pans, the carriage wheels and the manner of linking the pan carriages are shown in Fig. 228. In order to facilitate breaking the cakes of matte the pans are made with a ridge in the center, while in order to reinforce the pans and also to aid in dissipating the heat, corrugations are cast on the under side of the pan body. The ridge in the pan bottom is eaten out first by the matte, and it is in the central part of the pan that the corrosive action concentrates once that eating of the ridge begins. In order to enable this central part to be renewed, sectionalized pans were experimented with. These consisted of three parts held together by bolts that ran lengthwise through the pan. Owing

to the expansion and cooling of the pan these bolts sheared quickly so that a considerable amount of inspection was necessary to keep the conveyor pans in running condition. To put in these bolts working around a hot conveyor is no easy task and proved expensive. Moreover, the joints between the sections offer good points of attack for the matte. Consequently, it has been decided that the solid pan shown in Fig. 228 is the better of the two. When not used for casting matte these conveyors are used for handling converter slag.

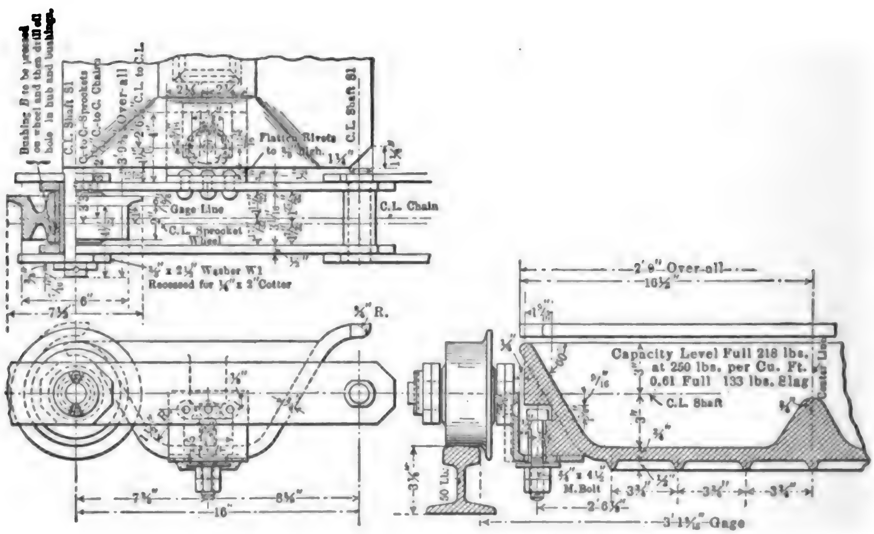


FIG. 228.—MATTE-CONVEYOR PAN, MAMMOTH SMELTERY.

Matte-pouring Crane.—In Fig. 229 are shown the details of the construction of the jib crane and the ladle cradle used at the Mammoth smeltery of the United States Smelting, Refining & Mining Co., at Kennett, Calif. Experience showed that it was advisable to use the power-operated cradle in pouring the matte, instead of holding the traveling crane in order to operate this cradle. The matte is poured from the ladle into a casting conveyor and is afterward resmelted in the blast-furnaces.

Double Trunnion Matte Ladle (By Charles F. Shelby).—In Fig. 230 is shown an improved type of matte ladle that was designed by me for the Cananea Consolidated Copper Co., by which this ladle has been adopted, as well as by the Cerro de Pasco Mining Co. and probably other smelters. One of the main points of interest in connection with this ladle is the employment of a double trunnion on which the ladle is suspended. Copper smelters have always experienced difficulties in exactly locating the trunnions on a matte ladle and if they should be placed so

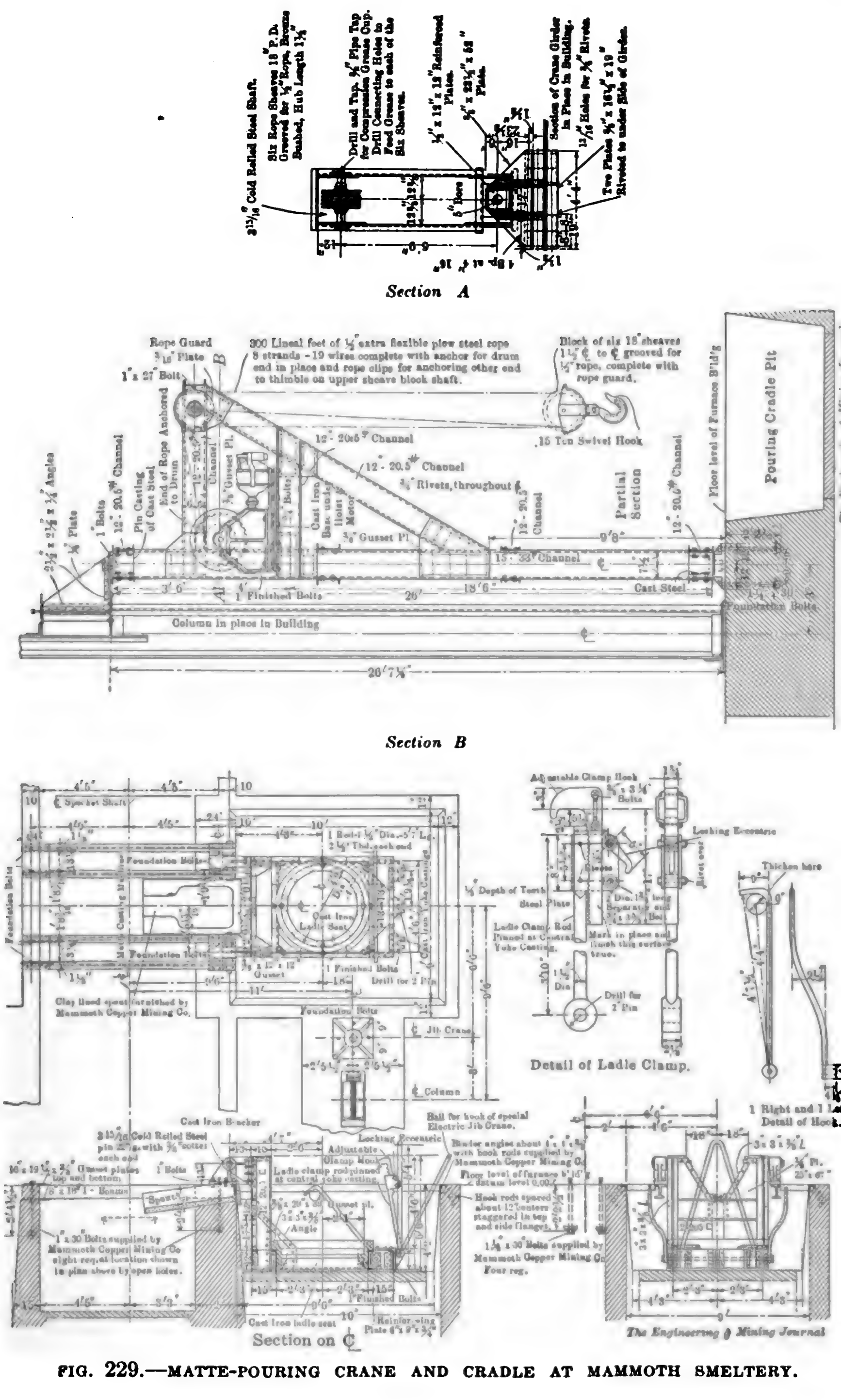


FIG. 229.—MATTE-POURING CRANE AND CRADLE AT MAMMOTH SMELTERY.

that the ladle would balance when filled to a certain point it would be out of balance when not exactly filled to the same level, on account of the overbalancing effect of the spout. This double trunnion has completely taken care of difficulties in this respect, and the ladle herewith illustrated can be depended upon to carry its load without any danger of spilling

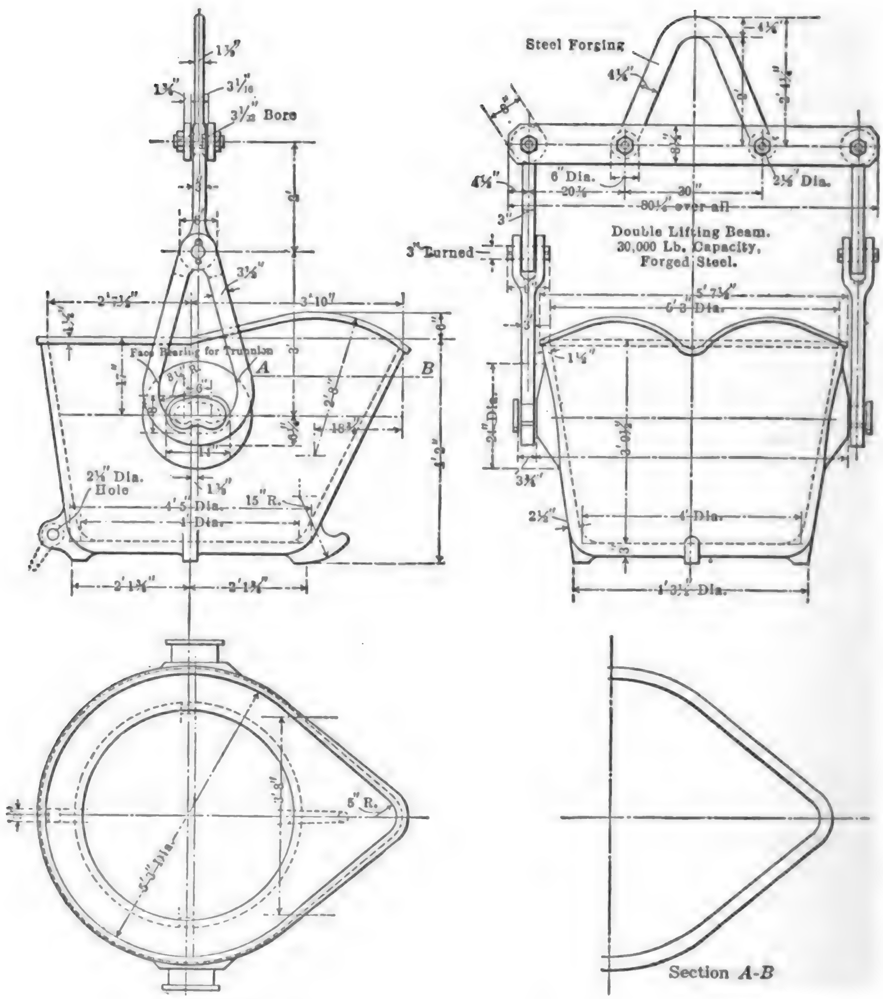


FIG. 230.—SHELBY DOUBLE TRUNNION MATTE LADLE.

by reason of being out of balance. It has a few other important features, among which might be mentioned the absence of any special lines of curvature in connection with the spout. This makes it easy for the skull to fall out when the ladle is turned upside down.

The Basic-lined Copper Converter.—Writing in the *Bulletin* of the American Institute of Mining Engineers, of June, 1913, E. P. Mathewson outlines the history of the adoption and development of the basic-lined converter, as applied to copper bessemerizing. He states that practically all the bessemerizing of copper matte in the United States to-day is done in basic-lined converters. The main points to be observed for successful operations are: Not to exceed a temperature of 2100° F.; not to have tuyeres smaller than 1¼ in., 1½ in. being the preferred size; to drive in punch rods the full size of the tuyere opening immediately after pouring copper; to maintain in the converter as large a mass of matte and slag as possible, in order to prevent sudden changes in temperature and overheating of the lining; and to employ a slag containing preferably about 25% of silica. A test made with a view to finding out whether the cutting action of converter slag on a magnesia brick lining bore any relation to the silica content of the slag was made, 13 slags being selected from daily samples sent to the laboratory and analyzed for silica and magnesia. The result showed no relation whatever between silica and magnesia content. The basic-lined process shows the following advantages: Decreased cost of lining; ability to use large units with resulting economies in labor, power and repairs; neatness and cleanliness of plant with consequent abolishment of danger to the health of the lining crew, arising from dust.

The Treatment of Overblown Charges in Copper Converters.—(By A. R. McKenzie).—Many disastrous results, some of which have proved fatal, have occurred on account of the incorrect treatment of overblown charges in the copper converter. One of the first methods that was generally used to treat an overblown charge, which by the way was once a common occurrence, was to bank the slag and oxide of copper floating on the surface of the charge, with dirt from the floor. As this was found to chill the charge of copper materially, it was discontinued, and the throwing in of large pieces of cold matte was substituted. The matte would slowly melt and react with the oxides of copper, lowering the pitch of the copper so that it could be poured into molds. Although this method gave fairly good results, the copper would be cooled to a considerable extent and accumulate on the mouth of the converter while the charge was being poured.

The next attempt to improve the treatment of overblown charges, was to tap a small pot of matte from one of the resmelting furnaces and add as much as was needed by means of hand ladles. The liquid matte would react almost immediately with the oxides, and after a few minutes the copper would be ready to pour. After the advent of the overhead electric crane, the problem was to a degree much simplified, but at the same time made more dangerous, from the standpoint of the inexperi-

enced, as the following reaction and explanation will show. To calculate the liberation of sulphur dioxide (SO_2), when liquid matte is added to an overblown charge of copper, let us for convenience assume that copper matte has the following composition—$Cu_2S.FeS$, then:

$$5CuO + Cu_2S{\cdot}FeS = 7Cu + FeO + 2SO_2$$
$$400 + 160 + 88 = 448 + 72 + 128$$

Thus we find that 248 oz. of matte ($Cu_2S.FeS$) reacting with 400 oz. of CuO gives off 2 volumes of SO_2 gas or 44.44 cu. ft. at 0° C. and 760-mm. pressure.

Correcting for temperature and pressure on the basis of atmospheric pressure at Great Falls, 677 mm. Hg., and assuming temperature in the converter as 1100° C., we have,

$$44.44 \times \frac{760}{677} \times \frac{1100 + 273}{273} = 251 \text{ cu. ft.}$$

Thus the addition of 248 oz. of hot matte (15.5 lb.) to an overblown charge of copper will set free 251 cu. ft. of SO_2 gas at 1100 ° C. and 677-mm. pressure or 16.2 cu. ft. per lb. of matte added. As ordinarily 1000 lb. of matte are used, this is sufficient to liberate 16,200 cu. ft. of gas.

This is the reason why extreme caution is needed when adding liquid matte to an overblown charge of copper, for, should the above reaction take place almost instantly, the liberation of 16,200 cu. ft. of sulphur dioxide would rend the converter, and throw the contents of it in all directions. Consequently a safe *modus operandi* in the treatment of overblown charges should be as follows:

Should a skimmer be unfamiliar with the conditions, he should notify his foreman immediately, who should then inform the craneman that the charge has been overblown. (As a matter of safety it would be well to have a written notice posted in the crane describing the proper course to pursue.) Pour the liquid matte into the converter in very small doses, 10 to 15 lb. at a time, backing up the crane each time, until the SO_2 gas stops issuing copiously from the mouth of the converter. These doses should continue until no more SO_2 gas is evolved, after which a sufficient amount of matte should be added to compensate for the loss of heat due to the endothermic reaction between the oxides and sulphides. This generally amounts to as much additional matte as was used to reduce the copper oxides.

The converter should then be turned up and the charge refinished, and it will be found that the copper is in the proper condition to pour.

The thermal reactions which take place are as follows:

First reaction:

$$5CuO + Cu_2S + FeS = 7\,Cu + FeO + 2SO_2$$
$$5 \times 80 + 16 \quad + 88 = 448 \quad + 72 \quad + 128$$
$$37.7 \times 5 + 20.3 + 24 = \ldots. + 65.7 + 138.5$$
$$-232.8 + 204 = -28.6 \text{ Cal.}$$

Thus it is seen that there is a loss of heat during the reaction; this together with the loss of heat due to radiation chills the charge in the converter. This loss of heat is compensated by the addition of the extra matte in excess of what is necessary to reduce the overblown copper.

Second reaction:

$$\begin{array}{l}
2(Cu_2S.FeS) + 5O_2 + SiO_2 = 4\,Cu + 4\,SO_2 + \frac{(FeO)_2}{SiO_2} \\
320 + 176 + 160 + 60 = 256 + 256 + 204 \\
2(20.3 + 24) + (..) + 180 = (..) + 4 \times 69.2 + 333.6 \\
- 268.6 + 610.4 = + 341.8
\end{array}$$

Or per molecular weight of $Cu_2S.FeS = 170.9$ Cal.

In copper plants where the copper is taken from the converters and put into refining furnaces it is not necessary to be so particular about treating overblown charges as they will do no harm, as the charge must be rabbled anyway; but where the copper is taken from the converters in ladles and poured at a casting machine, as is the common practice in most plants, it is essential to have the copper hot and free from slag in order to prevent the excessive chilling of the copper in the ladles, and the subsequent breaking up of the skull and the resmelting of the same.

Water in Converter Air Mains a Source of Danger (By A. R. McKenzie).—That all atmospheric air contains a certain amount of moisture is a well-known fact. The quantity of this moisture per unit volume varies with the locality, the seasons, barometric pressure, temperature, etc. The higher the temperature of the air, the greater the capacity it has for carrying moisture in the form of vapor, for example: At 32° F., saturated air carries 2.13 grains of water per cu. ft.; at 50°, 4.10 grains; at 60°, 5.77 grains; at 70°, 8.01 grains; at 80°, 10.98 grains; at 90°, 14.85 grains; at 100 °, 19.84 grains. Consequently if air saturated with water vapor at a higher temperature is compressed, and while being transported for any distance, has its temperature materially lowered, there is bound to be a condensation of moisture. In the compression of air for converter work, under conditions as above, there is an element of danger presented.

During my experience in copper converting, at least two glaring instances of this phenomenon have come to my notice. An exaggerated case of this occurred about 4 years ago, at a smeltery situated in a high altitude. The power house at this plant was about 1000 ft. away and situated on the same level as the blast-furnace feed floor. From the power house the converter air main passed through a tunnel in the brow of the terrace to within 70 ft. of the furnace building, whence it passed diagonally across to the outside of the converter-crane track, where it entered the converter building; from there it was curved downward to the level of the mouth of the converters, when they were in an upright position in the stalls. Before reaching the converters the pipe was en-

larged to a diameter of about 5 ft., which answered the purpose of a receiver or a reservoir. The connection to each converter was made from the side of this reservoir, which ran along behind the converter smoke boxes.

One rainy day about 2 o'clock in the afternoon, 4 months after the plant had started operation, I noticed water coming out of some tuyeres that were leaking air around the ball valves. I immediately had all the converters shut down, in order to ascertain the source of this water. On account of the rainy weather, my first supposition was that the water had got to the intake of the compressor and was being pumped to the converters with the air, but I soon discovered that such was not the case. I next made an examination of the 5-ft. reservoir by tapping it with a hammer. From the sound I judged it was full of water up to the outlets, where the side connections were made to the converters. A scaffold was hurriedly built and an air-power drill set in place. In a few minutes a stream of water was running from an inch hole in the bottom of the reservoir, the total amount of water drained from this pipe amounting to about 100 bbl. Afterward I had this hole fitted with a 1-in. drain pipe, with a valve which was kept slightly open to prevent a recurrence of what had taken place.

In the design of converter plants, care should be taken not to tap the converter air main from the bottom, for in case one of the converter stalls should be shut down for any length of time, the air pipe leading from the air main to the stall might fill with water, and in again putting this stall in operation, there would be danger of an explosion unless the collected water were removed. The difference in temperature between a hot engine room, where a number of machines are running, and the outside atmosphere, is so great, especially in cold weather, that condensation of moisture in the pipes goes on rapidly, and may become a source of danger in any case unless special means of drainage are provided. It may be thought that, in plants in continuous operation, enough water would never come through the main to cause an explosion, even without any special means of drainage, but the following case may be assumed to show such danger exists.

Suppose that one or two converters have been running for some time, and that some condensation of water has taken place in the air main. Then, should the number of converters be increased to four or five, as frequently happens in meeting changing requirements of production, etc., the volume of the air passing per minute will be perhaps doubled, tending to form waves and to sweep it into the connecting pipes, whence it will pass into the converters and cause an explosion. During intermittent operation there is no question of the danger. Some years ago, at another plant at which I was employed, a stall which had long been

idle was again put into service. The converter was connected in place and charged, without examination of the piping, which had slowly collected condensation water back of the valve. On turning into the stack and putting on the air, this water was carried into the tuyeres and blew the converter bottom to pieces, illustrating the extreme danger of allowing water to collect in the air system.

Movable Converter Hood.—At the Butte Reduction Works the converter gases were carried off through a movable converter hood, designed by James Doull, and shown in Fig. 231. The hood is mounted on wheels and is operated by an air cylinder having a 6-ft. stroke. When in blowing position, the hood is well over the mouth of the converter, forming an efficient outlet for the gases and reducing the escape of the converter fumes to a minimum. The gases pass into the converter flue from the top of the hood, near the back end, where the connection is so close that with ordinary suction in the flue no gas can escape. Nevertheless there

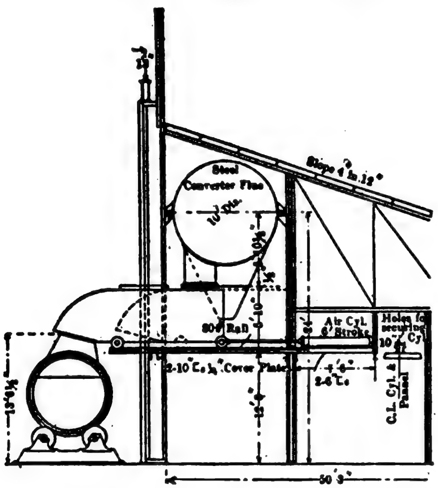


FIG. 231.—MOVABLE CONVERTER HOOD.

is no interference with the horizontal movement of the hood when it is desired to change the converter shell. With the converter hood under easy control of the operator, the stringent converter gases are kept out of the building and not only is the comfort of the converter men augmented, but when furnaces are in the same building more efficient work in furnace tending can be counted upon by reason of this elimination of the converter gases. In many of the older plants where stationary or unwieldy hoods are employed it is frequently impossible for men to remain at their stations when unfavorable winds carry the escaping converter gases through the building thus rendering crane service nearly impossible.

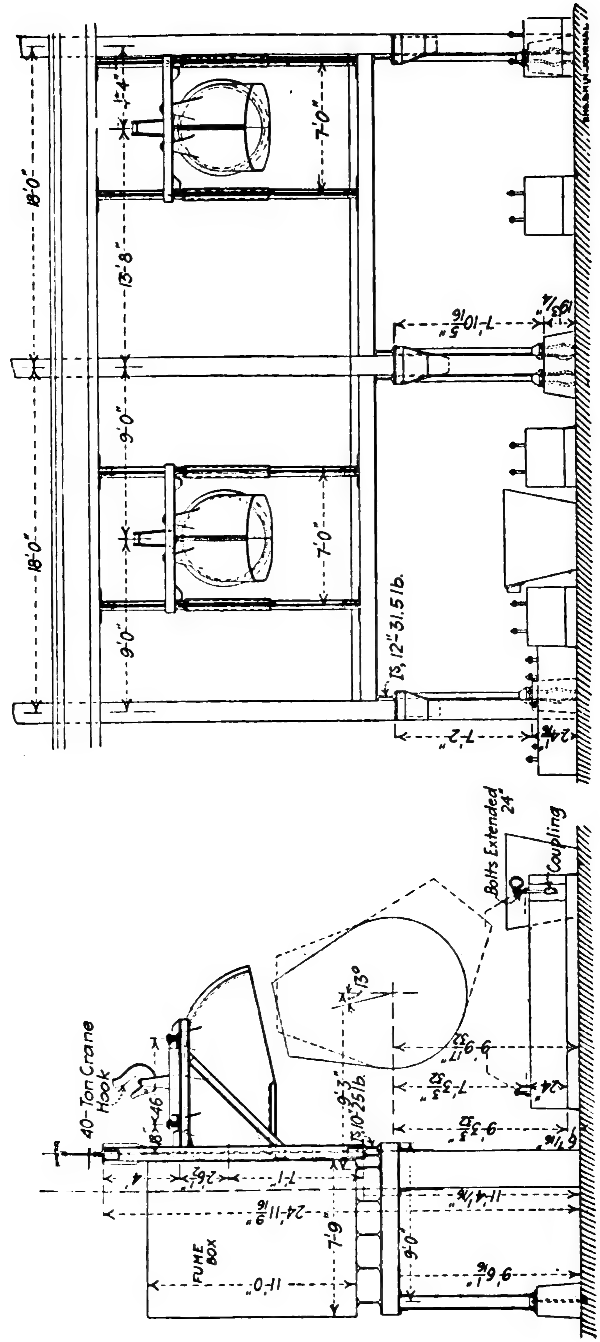


FIG. 232.—GENERAL VIEW OF BROWER CONVERTER HOOD.

The Brower Converter Hood (By Richard H. Vail).—The converter hood designed by C. L. Brower, of Chrome, N. J., was briefly referred to in the article on the "Copper Smeltery of the U. S. Metals Refining Co." in the *Eng. and Min. Jour.* of May 24, 1913. A photograph of the hood in operating position was shown in the original article, but the construction will be more readily understood from the accompanying detail drawings, Figs. 232 and 233. The hood proper is simply two steel castings bolted together, to which is attached the suspending frame. The latter is built up mainly of 6-in. channels arranged both to rest on the knee-brace supports of the building columns, and to provide for suspension of the crane. When it is desired to clean the hood, the crane merely lifts it off the knee-brace supports and places it on the ground where it can be cleaned easily and completely when occasion permits. This is in marked contrast with the half-hearted cleanings that converter hoods usually receive, while a shell is being changed or is temporarily turned down. A spare hood of the Brower type can be placed in position in about the time the men would consume in getting started at cleaning a hood of the old type. The Brower hood when in position has no permanent connection with the dust box; the flanges of the hood simply abut on the front wall of the dust chamber where is cut a hole corresponding with the internal diameter of the hood. The circular section of the Brower hood would seem to be better adapted to carrying away the gases from the converter mouth than the common hood of rectangular section, and has the further advantage of being more nearly over the mouth of the converter so that it seems to receive and direct the gas stream without seriously reducing the velocity or permitting the "backwash" so common with hoods of rectangular cross-section. In this respect, it is similar to the movable hood designed by James Doull for the Butte Reduction Works, but the Brower hood has the advantage of being more simple in construction and of being removable to the ground for cleaning. The converter room at Chrome is exceptionally free from smoke leakage either at the hood mouth or at the junction with the dust box.

OTHER METALS

Cleaning Dwight & Lloyd Grates.—At one of the lead-smelting works where Dwight & Lloyd sinterers are used it was formerly necessary to keep a man at each machine to punch open the slots of the grates. This man was eliminated by casting the grates with the slots parallel with the longitudinal axis of the machine, this permitting the installation of a simple mechanical cleaner. The latter consists of a series of ordinary iron washers, strung on a rod with intervening separators, so that each washer registers with a slot in the grate. The rod carrying these washers

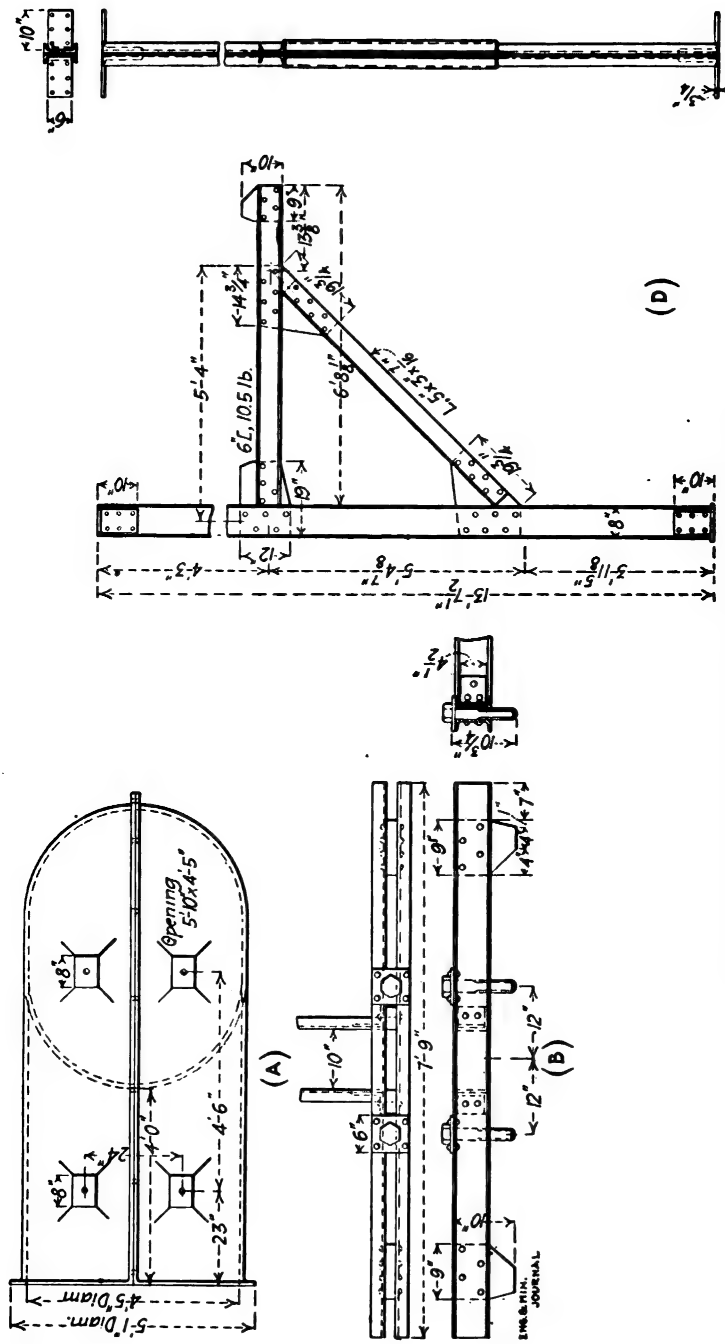


FIG. 233.—DETAILS OF THE BROWER CONVERTER HOOD.

is hinged at the ends. The movement of the grate lifts the rod with its washers and causes it to fall in one row of slots which are thus effectually cleaned. On the next movement of the grate the washers are lifted and on the next they fall into the next row of slots, all the movements being suitably timed, of course. This device was an invention in the works. A patent has been applied for.

Zinc-furnace Shield.—A shield for the front of zinc-distillation furnaces, as used by the Société Anonyme G. Dumont & Frères, in Sclaigneaux, Belgium, is shown in Fig. 234. The pivoted slats, which are hollow, have openings in order to permit the circulation of air through them.

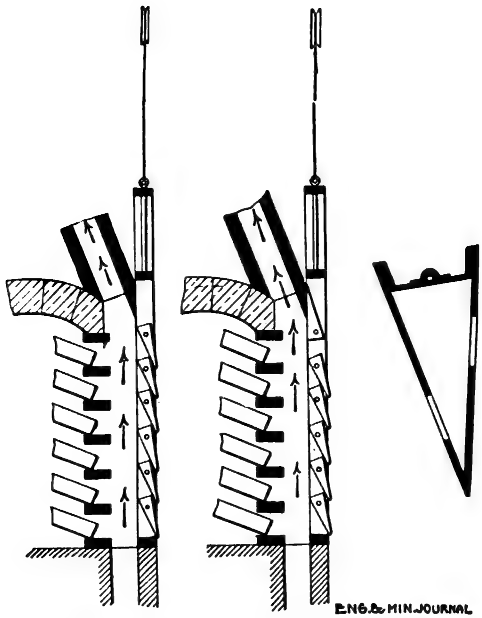


FIG. 234.—SHIELD FOR ZINC FURNACES.

Laying a Magnesia Furnace Bottom.—The accompanying description of the furnace bottom and method of laying same in the 15-ton electric steel furnace at South Chicago was given at a meeting of the Chicago section of the American Electrochemical Society. With small changes the method is available for non-ferrous furnace bottoms. The furnace shell is of plate steel, 1 in. in thickness, riveted together. The outside horizontal cross-section plan is approximately that of a complete circle 13½ ft. in diameter, with two flattened portions situated at the front and back respectively. On the botton of the furnace, within the 1-in. plate, and next to it, one row of magnesite brick laid the 4½-in. way is placed across the flat portion. The side walls of the furnace are vertical

and consist of two rows of magnesite brick laid the 9-in. way, giving a thickness of 18 in. of magnesite brick. These solid magnesite brick walls extend up to the furnace roof. The bottom proper of the furnace consists of dead burned Spaeter magnesite to a depth of 12 in. at its thinnest point, which is, of course, at the extreme center. From this thinnest point the bottom slopes gradually upward so as to form a portion of a sphere 7 ft. 2 in. in radius. This bottom was fixed in the following manner: Dead burned and carefully ground Spaeter magnesite was mixed with basic open-hearth slag in the proportion of four of magnesite to one of open-hearth slag. To this mixture enough tar was added to make the mass sufficiently plastic to enable it to be tamped into the furnace in the usual manner. The entire depth of the bottom was tamped in this way. Next the furnace was filled with wood, dried out for about 48 hr., and then filled with coke and the electrodes lowered and the current turned on. In this way the bottom was fluxed into place.

VIII

REFINING

TREATMENT OF AMALGAM

Acid Treatment of Mercury at Florence-Goldfield Mill.—It has been found at the Florence-Goldfield mill that it is good practice to distil the quicksilver into dilute sulphuric acid, as the acid attacks any distilled impurities such as zinc and lead which may have gotten into the amalgam. This acid dissolves them and holds them in solution, thus preventing them from contaminating the mercury. Mercury distilled in this manner is more lively than even new quicksilver. The sulphuric acid does not attack the mercury to any appreciable extent. In making cleanups at this mill the amalgam from the batteries and the plates is ground in a small grinding barrel with sodium peroxide. The sodium peroxide attacks the base metals in the amalgam and oxidizes them so that they cannot again go into the amalgam. In this way the bullion from the cleanups has been made to run 997.6 fine, while formerly it rarely ran over 930 fine.

A Batea and Amalgam Press.—Several devices have been arranged for cleaning and squeezing the amalgam taken from the copper plates of gold mills, wherein the pulp is subjected to amalgamation at any stage in the process of ore treatment. Perhaps the most common device for softening and cleaning this amalgam is the cylindrical amalgam barrel. In Australia the Berdan pan is in great favor for this work, and is also used in many mines in the United States, such as the North Star in California. In the gold mines of the Rand, the batea is sometimes used for cleaning amalgam, but it is not frequently seen in the gold mines of the United States. The batea there in use is described by C. O. Schmitt, in "A Textbook of Rand Metallurgical Practice, Vol. II," and is illustrated in Fig. 235. The only feature calling for remark about this batea is the rim of the pan which is made rather higher than usual. The motion of the batea is obtained from a crank driven through a set of bevel gears by a countershaft. The usual speed of the crank is 125 r.p.m. It is usual to suspend the batea as shown in the illustration. After softening and cleaning the amalgam from the plates, either in an amalgam barrel, Berdan pan, or batea, some sort of press is necessary for squeezing out the excess mercury. This is also illustrated in the drawing.

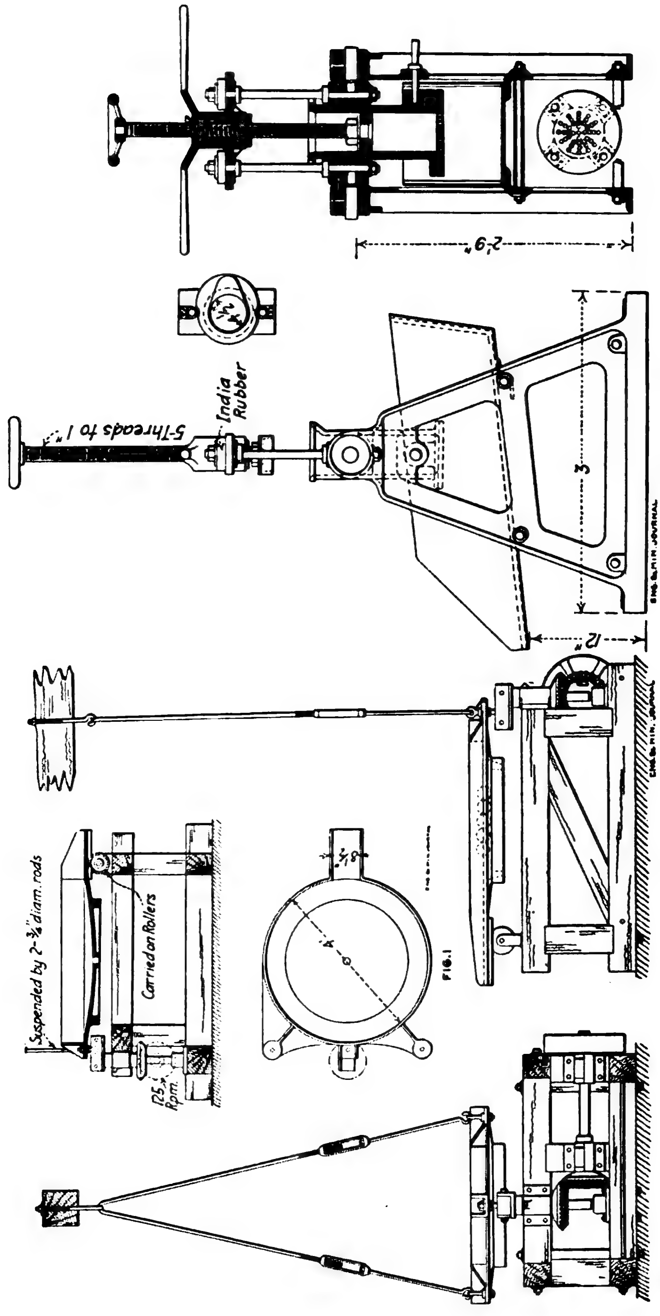


FIG. 235.—END AND SIDE ELEVATION OF BATEA AND AMALGAM PRESS USED ON THE RAND.

A Small Cleanup Mortar.—In Fig. 236 is shown the details of a mortar and muller which are used for cleaning up small quantities of amalgam from gold-milling operations. It consists of an ordinary mortar fixed to a wooden table and mounted on a shaft so that it can be tilted into a sink and emptied. The muller is power driven and arranged so that it may be raised and lowered, with respect to the mortar, by means of a lever as shown. A hook on the lever handle allows the muller to be suspended free of the mortar when desired. The device was designed by Mr. Schmal, superintendent of the South Eureka mill, Sutter Creek, Calif., where it is in use.

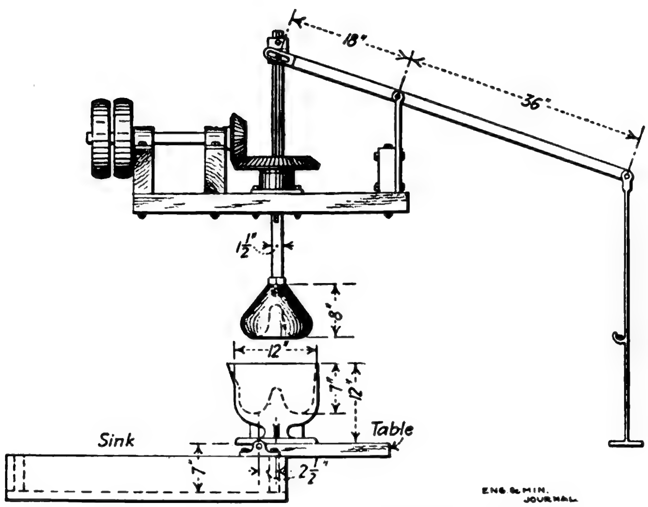


FIG. 236.—CLEAN-UP MORTAR FOR SMALL PLANTS.

Explosions of Amalgamating Barrels.—At a meeting of the Chemical, Metallurgical and Mining Society of South Africa, attention was drawn to the danger attending the use of barrels for grinding black sand and other rich products in mill cleanup rooms. At times gas is generated during their running which escapes with violence when the barrel is opened. A peculiar accident was reported in which a screw plug was used shorter than the thickness of the barrel. The dead space filled with a sticky black mixture during the grinding. Afterward the attendant placed some lime on this and then attempted to break the crust in order to introduce the lime into the barrel. The gas rushed out through the break, throwing lime into the man's eye, causing its loss. Another member reported trying to investigate the contents of a newly opened amalgamating barrel, trying to use a naked light to see by, with the result that he caused a severe explosion. Experiments seemed to show that the effect is due to small quantities of sulphuric acid being formed by oxida-

tion of pyrite and that this acid acts on the iron of the balls, etc., producing hydrogen. Another suggested reaction is that some of the iron and pyrite react to form ferrous sulphide, and that sulphuric acid, formed as above, acts on the ferrous sulphide to fórm sulphuretted hydrogen. The effects can be prevented by adding lime or other alkalis before grinding. It was also suggested that where it was necessary to examine the inside of a barrel with a naked light, that it was always wise first to use a jet of compressed air to blow out the gases in the barrel.

Mexican Method of Retorting Amalgam (By A. M. Merton).—I have never seen described the remarkably simple but effective Mexican method of retorting amalgam obtained from the operation of *arrastres* or *tahonas*, as they are almost universally called in Mexico and Central America. The method of retorting used by the small operators in Mexcio requires no iron retort. Everything is home-made, but the results are fully as satisfactory as those obtained from the use of the small retorts obtained from supply houses. These little wrought-iron retorts warp out of shape after being used once or twice and the top fits wretchedly. The Mexican appliance, which of course never fits, even at first, is always the same and will last several burnings before breaking. It can then be replaced at a maximum cost of fifty *centavos*.

Several years ago I had a group of claims in southern Mexico in which a small body of rich ore was discovered. I decided to test this ground by treating the ore in *arrastres*. Incidentally I hoped to get a little revenue. I had in my scrapbooks a number of articles on the *arrastre*, but face to face with the problem of building one I decided that most of the accounts had been written by amateurs who had never actually built and run such a machine. I, therefore, engaged a native who had a local reputation as a *tahonista* to build and run a pair of 12-ft. *tahonas*. The construction of two really good appliances was completed in a week by the *tahonista* and a helper. The only tools used were an axe and an auger. Not a particle of iron entered the mechanism and the two completed machines cost something less than 25 pesos.

After about 10 days' operation of the *tahonas*, two handsome balls of gold amalgam were obtained. The *tahonista* suggested that he should "burn" the amalgam and recover the quicksilver. We had no retort but that did not worry José María. His wife, he explained, was a *lozera* of ability, that is, she could make good *ollas* and such pottery. Being curious to see the native method of retorting, as well as desirous of having the gold in pure form, I consented.

The *tahonista's* wife made two *ollas*, one somewhat smaller so that it could fit into the mouth of the other. The smaller of the *ollas* was used as the retort proper, the ball of amalgam being placed within it and wedged tight against the bottom of the vessel with several pieces of old

pottery. The larger *olla* was about half filled with water and was used as a condenser. A hole was dug in the ground deep enough so that when the large *olla* was buried the mouth was a few inches below the ground level. The small *olla* was placed mouth downward in the larger, the water being about an inch below the mouth of the retort. The sketch, Fig. 237, shows the general arrangement. Earth was filled in the hole and when buried, the bottom of the small *olla* projected a few inches above the ground. A dab of stiff clay was stuck on the bottom of the retort, to protect the fragile unglazed earthenware against a chance blow, and

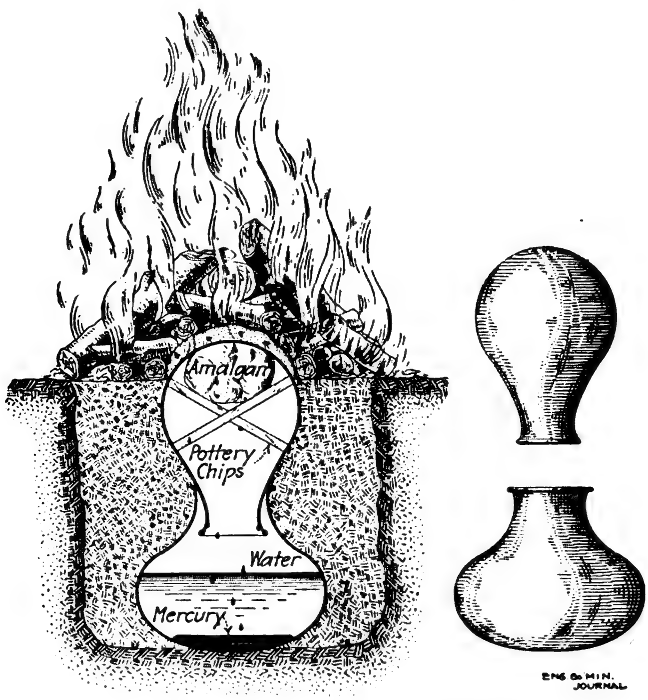


FIG. 237.—PRIMITIVE MEXICAN RETORT.

a fire was built over the hole. It was kept going briskly all day. In the evening the fire was raked away and the *ollas* taken up. In the small *olla* was as fine a retort of gold as ever I saw and absolutely free from mercury. In the large *olla* was the quicksilver.

Out of 124½ oz. of amalgam there was obtained 35¼ oz. of spongy gold and 76 oz. of quicksilver, indicating a recovery of about 84% of the latter. Certainly not a bad result for such a simple and apparently crude arrangement. The balls of sponge gold were melted down in a blacksmith's forge and gave a pretty bar which realized about 1100 *pesos*. The thought was thrust upon me that few indeed of our gold miners

could have accomplished the result with only the tools nature had supplied them as effectively as the illiterate Mexican *tahonista*.

An odd little trick of retorting amalgam was shown me by an old Guatemalan placer miner. We had tested some placer ground with the batea using mercury. To show up the pure gold the placer miner took a large potato, cut it in two, and in one of the halves gouged out a small hole which would comfortably accommodate one of the balls of amalgam. The two halves were wired together and the potato was placed in the ashes of a fire and slowly baked overnight. In the morning the potato, on being taken out of the ashes, was found to have absorbed the mercury, leaving the spongy gold in the cavity. The spongy placer gold made pretty and queer-looking beads.

MELTING AND REFINING PRECIPITATE

Handling Cyanide Precipitate.—Crushing and fluxing dried gold and silver precipitate is expedited, and dusting and chance for loss minimized by charging fluxes and precipitate together into a mixing barrel. This may be made of an oil barrel of wood or iron, a whisky barrel, or even a beer keg by providing a bolted hand-hole cover for charging and discharging. The barrel can be rotated by being placed loose on two rollers, one of which is power driven; or simple gudgeons, consisting of a 6-in. length of 2-in. pipe with a pipe flange, can be attached to either end of the barrel. For power driving, one nipple can be made long enough to receive a pulley, or the belt may be placed directly around the barrel. A few short sections of pipe, some pebbles or just scrap iron will assist in breaking lumps of precipitate. A thorough mixing which expedites melting will be effected in a few minutes. If it is desired to add niter, this should be mixed in coarse lumps after the fluxed precipitate is withdrawn from the barrel. A mixture containing a considerable quantity of short zinc and niter, thoroughly ground together, has exploded with a mild, prolonged roar while being heated in the crucible, lifting the furnace cover and discharging the contents of the crucible into the melting room.

Refining Zinc-box Precipitate (By Wilton E. Darrow).—For a number of years I used to smelt the acid-treated precipitate in the ordinary way and blow the impure bullion up with an air jet. Air under about 10 to 15 lb. pressure was introduced into the furnace by means of a section of ½-in. hose terminating in a piece of ¼-in. pipe about 4 ft. in length. The air line from the compressor comes in along the ceiling of the melting room, and the hose and ¼-in. pipe nozzle were adjusted to enter the melting pot through a notch cut in one side of the crucible cover. By this means the air jet impinged on the surface of

the molten metal at an angle of about 45 deg. The point of the pipe was kept about 4 in. from the surface of the metal, so as to diffuse oyer about one-half of the surface. In this way I could blow up about 15 lb. of speiss and lead in the bullion in an hour. The fumes, however, were extremely unpleasant. The slag produced was also exceedingly corrosive and rapidly destroyed the plumbago crucibles. The greatest objection, however, was that the slag was rich, containing on an average about 4% of the gold and about 10% of the silver.

About 6 months ago, after making a few laboratory tests, I installed a small chlorination plant for refining the precipitate. In my laboratory tests I found that I could extract about 999½ parts in a thousand of the gold from the acid-treated precipitate by treating it with a strong chlorine solution, and that after the chlorine treatment nearly all of the remaining gold and the silver chloride was soluble in cyanide solution. The small chlorine plant generates the gas in a small tub by means of sulphuric acid, salt and potassium permanganate. The acid-treated precipitate is placed in the tub and allowed to remain in the chlorine solution over night. The solution is then decanted off and the residue given several washes with clear water to remove the dissolved gold. The decanted solution and wash water are flowed to a larger tub set on a lower level in the refinery. When the gold solution has been transferred to the second tub, it is precipitated with iron sulphate solution and allowed to settle for 48 hr. before the waste solution is siphoned off.

As soon as the gold has been washed out from the original residue, the acidity is neutralized with caustic soda and cyanide solution added for the recovery of the remaining gold and the silver. The solution is agitated with an air blast introduced through a pipe and hose for several hours and then siphoned off slowly into a couple of small iron zinc boxes and the treatment repeated several times during the two days before the final cleanup of gold precipitate. The silver precipitate from the zinc boxes and the gold precipitate from the precipitation tub are then cleaned up and melted together, bullion about 960 fine resulting. The recovery from the original precipitate averages about 99½% of the gold and silver value.

By the former method it took about 16 hr., with two furnaces using No. 25 plumbago crucibles, to melt and refine the gold-silver bar. Now I do it in about 4 hr., with one melting pot. I think that the process is good, but the installation of a small lead-lined chlorinating barrel for dissolving the gold would be an improvement. Perhaps some metallurgists would go a step farther and recover the gold by electrolysis, thus dispensing altogether with the disagreable melting process.

High-grade Bullion from Precipitates[1] (By J. Boyd Aarons and Herbert Black).—In order to find the best method of producing high-grade bullion from zinc-box precipitates extensive experiments were carried on at the Chaffers gold mine, Western Australia. The usual procedure followed at cleanup is to send all the precipitate and short zinc to the acid vat, only the well-washed long zinc being returned to the boxes. No scrubbing or screening of short zinc is attempted, it being desired to obtain the maximum amount of gold possible. After acid treatment, the finishing point being determined by the absence of action on the addition of acid, the diluted precipitate is sent to a small filter press where the pressed precipitate is given three or four washes of hot water in order to wash out the soluble sulphate. The cakes are air dried and dropped into shallow iron trays, weighed, and placed into cast-iron muffles, where the precipitate is subjected to either a roast at dull red heat or is merely dried. No screening of the dried or roasted precipitate is carried out, as smelting operations prove that the practice of screening sometimes adopted has no bearing on the production of high-grade bullion.

The roasted or dried precipitate after being weighed and sampled is mixed with flux as follows: 100 parts precipitate, 50 parts borax, 20 parts sand, and in some cases 7 to 10% of niter is added to the charge. This mixture is melted in clay liners in plumbago crucibles (No. 100). When fusion is quiescent the charge is poured into conical molds producing high-grade buttons of gold bullion averaging about £4 4s. ($20.40) per oz., which are subsequently melted in small plumbago crucibles (No. 30), and the gold poured into bar molds of 500 oz. capacity.

An investigation of the physical nature of the roasted or dried precipitate and of the bullion under consideration makes it plain that it is easier to produce high-grade bullion direct from the roasted or dried precipitate than to produce low-grade bullion and afterward refine it; moreover, there is less chance of loss where the refining is carried out through the medium of the flux than in the refining of the metal.

Some authorities are emphatic on the necessity of a dull-red heat for roasting the precipitate before smelting operations. In the event of the acid treatment being imperfect, either owing to insufficient acid, or to a protective coating of insoluble lead or calcium sulphate deposited on the zinc, a roast would be necessary, or the use of clay liners and niter flux would have to be resorted to in order to obtain high-grade bullion.

The object of roasting at dull-red heat is to oxidize any metallic zinc present, but roasting opens an avenue for loss of gold through the gold being carried over with the copious fumes of zinc; therefore, it is

[1] Abstract of an article in *Monthly Journal* of the Chamber of Mines of Western Australia, Aug. 31, 1911.

advisable to give a thorough preliminary acid treatment and thus avoid the necessity for roasting at red heat. Provided that the acid treatment be practically perfect, the drying of the precipitate at 100° C. and subsequent melting in clay liners and plumbago pots is all that is necessary to obtain bullion of high grade.

With regard to base metals present in the precipitate under consideration, the only troublesome base is apparently zinc, which is dealt with as above. Practically no copper is present and proof of this is seen by the high-grade bullion produced by the use of liners without niter flux. Were copper present in this precipitate, then the fine filaments of this base metal would remain unchanged throughout the acid and wash treatment and metallic copper would come down in the bullion unless precaution were taken for the elimination of this base. The use of clay liners and niter flux would result in high-grade bullion. Experience has proved that the precipitate containing a large percentage of copper which in plumbago crucibles only gave bullion worth £2 2s. ($10.20) per oz. was raised to £4 3s. ($20.16) per oz. by the above method. The practice of washing zinc shavings in a solution of lead acetate often accounts for the presence of lead in the bullion. This base may be removed by liners in plumbago crucibles using niter as flux but the method is slow and tedious. Manganese dioxide is often added to facilitate operations, and is preferable, because the elimination of the lead is performed more quickly.

Treatment of Precipitate at Waihi Mill.—The zinc-box precipitate from the three plants of the Waihi Gold Mining Co., Ltd., is brought into one melthouse, situated at the Waihi mill. The coarsest of the zinc slime is treated with sulphuric acid. Surplus moisture is taken out in a vacuum vat, and the wet slime is mixed with 20 to 25% borax glass and 10% soda ash. A small set of rolls is used to mix the flux and slime, which is then transferred to light iron trays holding about 100 lb.; these are put into a drying oven. The dried slime cakes firmly and is roughly broken to a convenient size for melting, which is done either in No. 200 plumbago crucibles (gas fired) or on a cupellation test. For the last two years[1] all melting has been done on the cupellation tests with coal fuel, but at present experiments are being made with pot melting by gas fuel, which promises economy.

Great difficulty was found in making tests that would stand the corrosive action of the slag during melting, and the action of the litharge during cupellation. A concrete shell with marble filling gave the best results. The gas for pot melting is conveyed from the mine producer-gas plant in a 3-in. pipe, and the cost at the melt-house is about 4c. per

[1] E. G. Banks, "Milling and Treatment at the Waihi Mine," *Min. and Eng. Rev.*, March 6, 1911.

burner hour (1000 cu. ft.). Two burners are necessary for each No. 200 crucible, and the slime can be melted at the rate of 40 lb. per hr. The slag is ladled from time to time, and the bullion is finally poured into 1000-oz. bars, which are about 940 fine in gold and silver. These bars are cupelled down to about 1% base—mostly copper—and cast into slabs 8 in. by 10 in. by ½ in., which are weighed, sampled, and sent to the refinery department. About 13,000 lb. of precipitate is handled each month, producing about 125,000 oz., the average value being about $3 per ounce.

The bars from the amalgam are refined with the cyanide bullion. The output of each mill and the concentrate plant is kept separate until sent to the refinery. All slags, litharge, and dross are run down in a 20-in. round water-jacketed blast-furnace, the lead bars being used for cupelling the next month's bullion, and the slag is crushed and passed over a Union vanner for recovery of metallics. The final slag tail usually assays from $3.50 to $7.50 per ton. The gases from the cupels and blast-furnace pass through long settling flues, and finally to condensing chambers, before escaping.

The Goldfield Consolidated Bullion Refinery.—The new refinery of the Goldfield Consolidated Mines Co. has many innovations; in effect, it is a small smelting plant that handles the products of the mill, and does away with the necessity of shipping concentrates and low-grade bullion. The precipitate from the filter presses is molded into small briquets, then dried in a baking oven. It is then smelted with lead in two small blast-furnaces. Silica and old cyanide tins are added to the charge for fluxing. The slag produced is so low in gold that it may be discarded. The base bullion is cupelled and bullion 0.900 fine is obtained. The products from the cupelling furnace, such as litharge, slag and old cupels are returned to the blast-furnaces for recovery of the gold contained.

Precipitate Melting at the New Belmont Mill, Tonopah (By A. H. Jones).—At the refinery of the new Belmont mill at Tonopah, there are two double-chamber No. 3 Rockwell furnaces which have satisfactorily melted 3,000,000 fine oz. of bullion, gold and silver, during the past 10 months. The cost of lining one of these furnaces is $231.45, divided into 25 sacks carborundum and kaolin, 2550 lb. at $0.04, $210; ½ bbl. water glass, $11.45, and $10 for labor. One lining has melted 500,000 oz., making the lining cost $0.462 per 1000 oz. The crucible cost with Faber du Faur furnaces, on No. 3 Dixon retorts, was $1 per 1000 oz. of bullion produced.

One melt of 105,676 oz. was melted in 36 hr. from the time of lighting the fires in the two furnaces, with the following cost: 1139 gal. fuel oil at $0.0383 per gal., $43.62; labor, $40.95, and power for blower, $6, making a total of $90.57, or $0.0009 per ounce.

The labor used included a head melter, two shifts at $175 per month,

\$11.70; melters, 2½ shifts, at \$4.50 per shift, \$11.25; helpers, 4½ shifts at \$4 per shift, \$18; making a total of \$40.95. The precipitate runs from 60 to 80% fine, and is melted without any acid or other treatment, the resulting bullion being over 900 fine in gold and silver.

Briquetting is done on a Grath Little Giant machine, furnished by Illinois Supply & Construction Co., of St. Louis, Mo. Two special disk-shaped dies were made, in place of the briquette die, 3½ in. in diameter, making briquettes from 2½ to 3 in. deep and, depending on pressure and feed, weighing about 2 lb. each. Running continuously, the machine will turn out two briquettes eight times per min.—or 1900 lb. per hr., but in practice the speed is about 1000 lb. dry product briquetted, together with required flux, per hr. The machine weighs 9230 lb. and cost \$997.50 plus \$410.74 freight, or a total of \$1408.24 delivered at Tonopah.

There are 80 ft. of 30-in. flue in the refinery, running from hoods 60 ft. outside of the house, raising at about 20°, with 20 ft. of stack. In cleaning the flue, about the same results are obtained each time. With 1,749,135.51 oz. melted, worth \$1,439,225, there was recovered from the flue by three cleanings, 124 lb. dust worth \$1 per lb. or 0.0086% of the bullion recovered. The analysis of the dust is given in Table XXXVII.

TABLE XXXVII.—ANALYSIS OF FLUE DUST

Insoluble	6.0%
Gold and silver	8.6
Si	1.4
Cu	0.5
Pb	Tr.
Fe_2O_3	2.4
CdO	8.2
ZnO	15.0
SO_3	14.0
Carbon and oxygen	Difference

Melting and Refining Precipitate at Tigre Mill.—At the Tigre mill, El Tigre, Sonora, Mex., the Merrill process of zinc-dust precipitation is used. The following description of the methods employed in melting and refining the precipitate is taken from an article by D. L. H. Forbes in the August, 1912, *Bulletin* of the American Institute of Mining Engineers.

The precipitate taken from the presses is dried in an electrically heated car, which has a series of superimposed shallow pans, under which the resistance coils are placed. On account of the large amount of moisture given off from the precipitate while drying, the resistance wires have to be protected by an asbestos paint covering in order to prevent corrosion. The car has a capacity for drying 1000 lb. of precipitate in 10 hr. and takes 4.5 kw. of power.

Melting is done in two No. 125 crucible Steele-Harvey oil-burning fur-

naces of the tilting type. The object of the melting is to fuse the precious metals and to remove as much of the copper and zinc as possible in a slag which will also carry off the baser impurities, such as alumina, silica, lime, and iron. To effect this purpose, fluxes are added to the precipitate in such proportions that the oxides of copper and zinc find sufficient finely powdered silica present to unite with it and form subsilicates; the oxides of iron, aluminum, and calcium find sufficient borax and silica to dissolve or unite with them; while a little sodium bicarbonate and fluorspar are added to make the slag fluid.

The precipitate analyzes as follows: Au, 0.3%; Ag, 63.13; Cu, 1.5; CuO, 3.14; Zn, 2.0; ZnO, 3.23; SiO_2, 11.6; Al_2O_3, 5.7; Fe_2O_3, 1.3%. For the melt 100 parts of precipitate are fluxed with 5.7 parts of silica, 5 parts of borax, 0.6 part of fluorspar and 9.1 parts of soda bicarbonate. When fluxed in this manner the precipitate melts down nicely. As the charge sinks in the crucible, fresh precipitate, mixed with flux, is added about every 20 min. until the crucible is nearly full of metal and slag in a state of quiet fusion. The slag is then poured into a conical pot, in which a crust is allowed to form to a thickness of 1 in.; the molten slag inside the crust is run out on cast-iron floor plates.

The crusts from this operation usually contain a small button of gold and silver at the point of the pot, and this button is broken off, and either put back in the crucible, or, if matte be present, is saved for a separate treatment with other similar buttons. The crusts themselves carry sufficient gold and silver to warrant a remelting, while the molten slag that is run on the floor is low in value, and, once a month, is weighed, sampled, and returned to the head of the mill.

After two to three pourings of slag, the molten metals in the crucible are subjected to a refining process, which depends upon the rapid formation of the oxides of zinc and copper when these metals are exposed to the air at high temperature in the molten state. With the furnace tilted forward, the operator throws a handful of bone ash over the surface of the molten metal. This thickens whatever slag still remains, and enables it to be easily removed by means of spirals of 0.5-in. round iron. The air striking the exposed surface of the metals rapidly oxidizes the zinc and copper. After a couple of minutes, borax is thrown on and dissolves the oxides, forming a slag, which is thickened with bone ash and removed as before. After about two repetitions of this refining, the precious metals are ready for pouring.

The furnace is tilted back and the heat applied for about 5 min. with the cover over the crucible. A step-bottomed truck, with bullion-molds that have been heated and greased with a mixture of oil and flake graphite, is then run in front of the furnace, the mold on the highest step being set in front of the spout. The bars of bullion are poured by

tilting the furnace forward over each of the molds in turn. After removal from the molds and cooling, the bars, which weigh from 1200 to 1300 oz., are chipped, sampled by drilling top and bottom, and stamped, ready for shipment. By this process, in spite of the high content in copper carried by the precipitate, bars from 850 to 900 fine are obtained.

From 20 to 30 lb. of flue dust are collected and removed each month from a large chamber outside the melting-room through which the furnace gases are led. A typical analysis of the flue dust is: Au, 0.01%; Ag, 21.39; Cu, 10.40; Zn, 7.50; SiO_2, 11.08; Al_2O_3, 2.90; Fe_2O_3, 2.30; undetermined, 44.42 (chiefly particles of borax, fluorspar, etc.); total, 100%.

For the remelting of slag crusts, scraps of iron box bands are added to the slag in the proportion of about 0.5% of the weight of the slag, and the slag put through the furnace until a button of sufficient size to collect in the mold is obtained. Slag assaying Au, 1.98; Ag, 650 oz. per ton; Cu, 7.5%; Zn, 13.5; SiO_2, 28.1; Al_2O_3, 11.34; Fe_2O_3, 4.76 was reduced in this manner to Au, 0.06 and Ag, 32.34 oz. per ton; yielding metal that assayed Au, 0.147; Ag, 40.183; and Cu, 42.4 per cent.

Precipitation and Refining, Yuanmi Mill, Western Australia.—At the new mill of the Yuanmi Gold Mines, Western Australia, the clarified solution is precipitated on zinc shavings in the usual manner (*Monthly Journal* of the Chamber of Mines of Western Australia, November, 1912). No especial care is necessary to get good results. The boxes are cleaned up once a month. Owing to the manner in which the gold plates the zinc, more short zinc is acid treated than is usually the case. But little free black slime is produced from these solutions; the zinc, when plated with gold, becomes hard and brittle. After acid treatment, the precipitate is roasted in a cast-iron muffle, and is fluxed with about 35% of borax and 5 to 10% of clean quartz sand. The bars are poured direct from a No. 9 tilting furnace into molds. The average value of the bullion produced (without any refining) is over £4 per oz. Clay-lined and ordinary graphite retorts have been tried and it is found that the latter yield bullion just as high in value as the former.

BULLION REFINERY NOTES

Refining Low-grade Bullion.—H. T. Durant has described a novel method of refining low-grade bullion. In one case of bullion assaying 250 fine, he granulated the metal in the ordinary way of pouring it into water. It was then mixed with half its weight of sulphur and heated rapidly in clay crucibles to the melting point of gold. The resulting bullion buttons, after breaking from the cooled crucibles and matte, were melted with iron and assayed over 850 fine. The matte was quite clean. Similarly, raw cyanide slime, acid treated but not roasted, was

fluxed as usual but with 3% of sulphur in addition. The bullion was 900 fine and the slag and matte were not unusually rich.

Handling Silver Nuggets at Crown Reserve Mill.—An interesting process for the production of bullion from silver nuggets has lately been evolved at the Crown Reserve mill at Cobalt, Ont. These nuggets, writes G. C. Bateman, are the ones that are left in the ball mill after the rest of the ore has been ground, and they are exceedingly rich in silver. It has been found cheaper to reduce these to bullion at the mine, than to ship them to the smelteries. To effect this they are placed in graphite crucibles in 400- to 500-lb. lots, and heated. No fluxes are used, but a current of hot air is forced from the blower over the top of the crucible. The waste is skimmed off, and the bullion is about 995 fine.

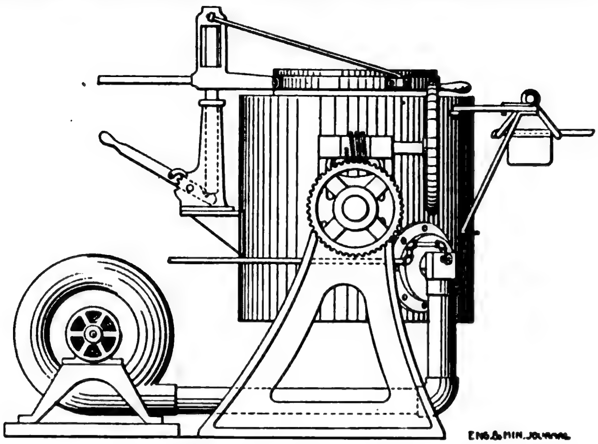
Electrolytic Refining of Silver Bismuth Alloys.—According to W. N. Lacey (*Trans.* Am. Electrochem. Soc., 1912, p. 301), laboratory ex periments showed that the best results were obtained with an electrolyte prepared by adding a 5% solution of potassium cyanide to a 5% solution of silver nitrate till the precipitate produced re-dissolved. A deposit containing less than 1% of bismuth may be obtained with a current density of about 0.14 amp. per sq. dm. The current density gradually decreases during electrolysis, and the potential difference between the electrodes gradually increases slightly, but it may safely be as high as 6 volts. The power required is about 0.25 kw.-hr. per lb. of silver at 1.5 volts, and increases by 0.11 kw.-hr. per volt increase in the resistance. The deposit is quite adherent and the use of anode bags is not necessary.

Starting-Sheet Preparation.—In making starting-sheets for electrolytic copper refining, the blank on which the light starting-sheet is deposited, and from which it is later stripped, should be placed in the electrolyte when current is passing. Neglect of this precaution may cause the deposited copper to "burn" on the blank, so that it is impossible to strip it off, and the blank is ruined.

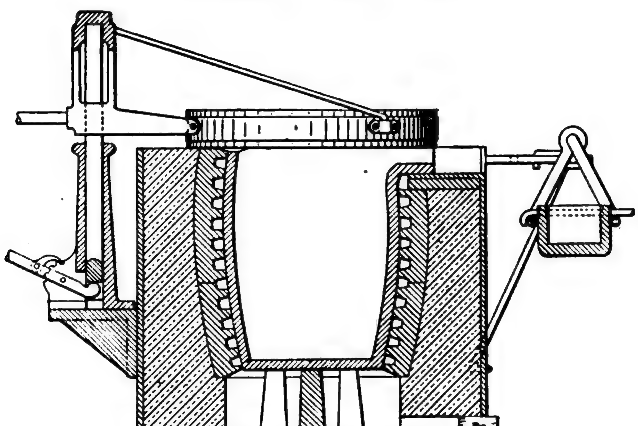
The Case Metallurgical Furnace.—A furnace for general metallurgical uses, such as melting precipitate, bullion and other metals, has been invented by W. W. Case, Jr., of Denver. The object of this new furnace, shown in Fig. 238, is to decrease the cost of operation, decrease loss by breakage of crucibles, and injury to the furnace lining and to obviate the usual roaring noise of such furnaces. The furnace is constructed, as is usual with devices of this kind, with an outer inclosing wall of refractory material, surrounded by a sheet-metal casing to hold the fire-proof wall together. The whole is carried on trunnions which rest in side supports, and one of these trunnions is provided with worm gearing operated by a hand wheel for tilting the furnace when pouring out its contents.

The interior of the furnace wall may be built as is shown in the sectional view presented, Fig. 238, the advantages of which are obvious.

The liner plates fit close around the crucible and are removable at will. The corrugations in these lining plates are not continuous but are arranged in a kind of checkerwork which does not oppose the passing of flame or heat about the crucible. At the top of the crucible a curved lip projects, meeting a corresponding depression in the top of the furnace wall which is designed to carry off the molten metal. The crucible bottom rests on supports, or legs, so arranged as not to obstruct the



Side elevation of complete device.



Section of furnace proper.

FIG. 238.—CASE METALLURGICAL FURNACE

flame passage. Flame for heating is delivered tangentially with the combustion chamber. A bullion mold may be suspended from the furnace in front of the lip. The oil burner is operated by a small blower delivering blast at about a 6-oz. pressure, thus avoiding noise.

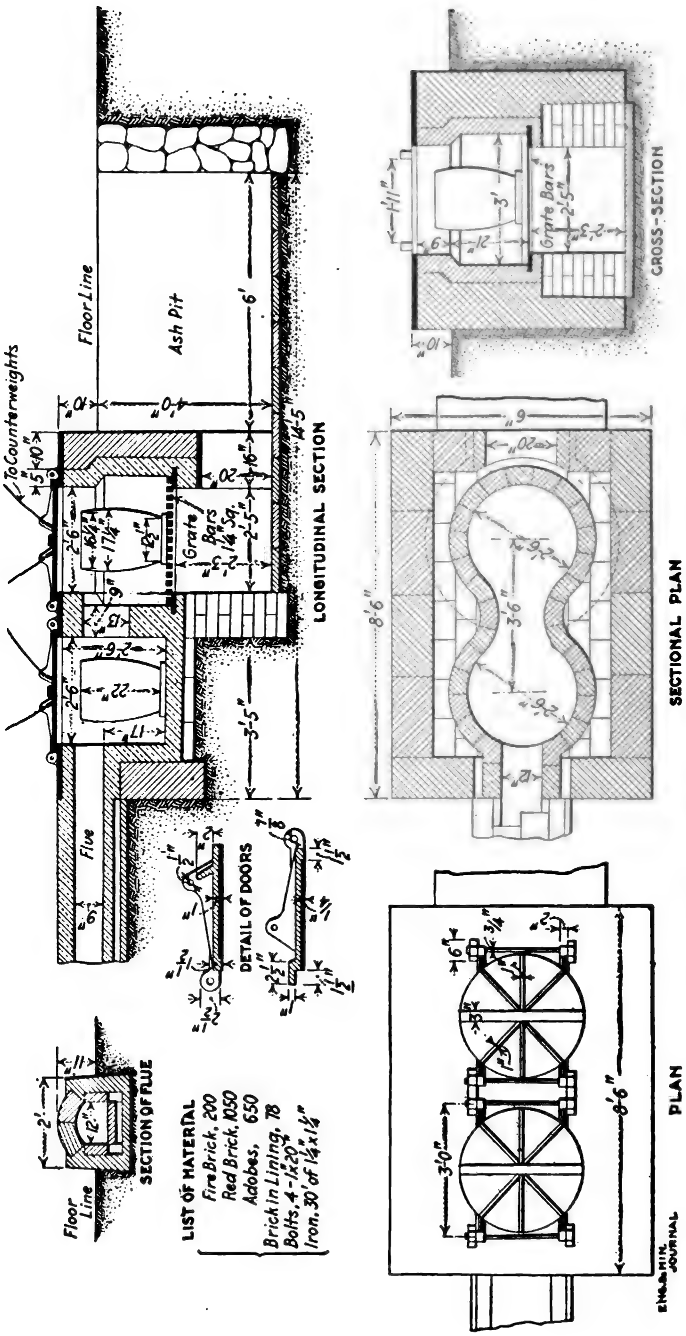


FIG. 239.—CHARCOAL-BURNING MELTING FURNACE USED AT RIO PLATA MILL.

Melting Furnace at Rio Plata Mill (By Alvin R. Kenner).—The furnace used for melting at the Rio Plata mill, Guazapares, Chihuahua, Mex., has several features to commend it over the usual type of pot furnace. This furnace originally contained four compartments, as shown in the top drawing in Fig. 240, and was installed by A. F. Hughes, a former metallurgist of the company. Only the first two compartments worked at all, and these not efficiently, consequently the old furnace was dismantled and the furnace shown in Fig. 239 constructed.

The inner lining is of firebrick, backed by a red brick made of a rather poor grade of local clay of highly silicious character. The rest of the furnace is made up of the usual Mexican adobe and topped with a 1½-in. iron plate. The drawing shows a change in the design of the doors, which was suggested by A. W. Prior, a former metallurgist and superintendent of the company, and which it was intended to introduce whenever new doors became necessary. However, the present doors are still in good condition and the old method of handling them still in use. A bar extends along back of the furnace at a suitable height to be used as a fulcrum for the long two-pronged lever used in removing the doors. The prongs of the rod fit into two eyes in the top of the door and the latter is readily thrown in and out of position.

In firing the furnace, a layer of about 3 in. of charcoal is maintained in the first compartment, which requires a shovelful of fuel every 5 to 10 min. This was found to be much more economical and gave a much quicker melt than filling up the furnace around the crucible. With a small amount of charcoal thorough combustion of the gases is obtained, which probably accounts for the better results. When the furnace is full it is difficult to obtain a sufficient amount of air even with a forced draft, and the caloric value of all the fuel is not utilized. The grate bars are 1¼ × 1¼ in. in cross-section, and are spaced ¾ in. apart. A No. 300 Dixon crucible, placed on a seat made of two pieces of square iron (1¼ × 1¼ in. in cross-section) joined by pieces of ⅝-in. rod iron at the ends, is used in each compartment.

The second crucible is usually ready and always poured before the first. When the doors of the second compartment are opened, the draft draws air through them up the flue and there being no grates in the second compartment, the heat to which the smelter is subjected is considerably reduced and an objection to the usual type of pot furnace greatly modified. In pouring the crucible from the first compartment, a shovelful or two of fresh charcoal is scattered over the fire and the draft shut off. There being no charcoal or slag matted around either crucible, the tongs can be clamped quickly and securely.

Each crucible when ready for pouring contains about 330 lb. of precipitate and flux. The capacity of the two compartments in 24 hr. is

about 18 bars of silver, each weighing from 1100 to 1200 oz. and assaying better than 980 fine, or a little over 18,000 oz. of pure silver per day. Attention should be called to the high capacity obtained by the method of firing employed. In a recent melt of 140,000 oz., the fuel consumption was less than 50 kg. of charcoal per 1000 oz. of bullion.

Many suggestions, based on the multiple-furnace idea as first installed by Mr. Hughes, have been offered as improvements on the furnace now used. These suggestions include many different arrangements of the baffle pieces, many different kinds of doors and several ways of reinforc-

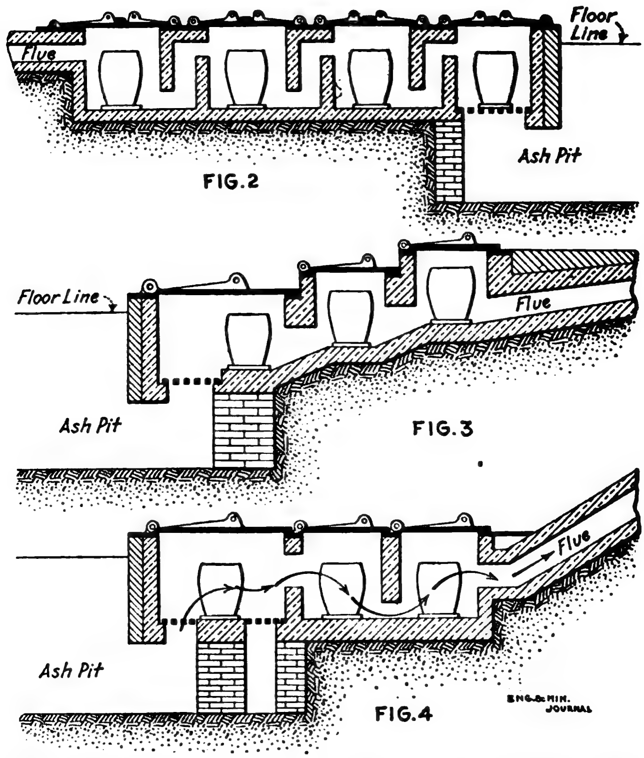


FIG. 240.—OLD TYPE OF MELTING FURNACE USED AT RIO PLATA MILL AND SUGGESTED MODIFICATIONS.

ing the grate bars. However, the tendency of these suggestions is to complicate the design, and simple construction is more desirable, both on account of cost and efficiency.

Two ideas which have never been suggested but which should add to the efficiency of the furnace now in use, are shown in Figs. 2 and 3 (referring to Fig. 240). It is evident from observation of the flame as it enters the flue of the present furnace that a third crucible could be used without materially increasing the consumption of charcoal if the design

in Fig. 2 were used and a slightly larger flue were constructed. The flue now in use consists of a 10-in. pipe, which is hardly large enough to create the proper draft and insure complete combustion. The obstruction, or wall, shown between the two crucibles in the longitudinal section of Fig.1, is of little practical use. In fact, it is a drawback in that it causes the flame to strike the crucible at the middle instead of the bottom.

The design in Fig. 2 overcomes this objection, as the flame will strike the lower part of the crucible and pass through the seat and underneath the crucible as well, in both the second and third compartments. In the first compartment, a hot flame is also obtained on the bottom of the crucible by placing the seat upon a solid base instead of upon the grate bars. More heat will strike the bottom of the pot so placed when resting upon an open seat than would be possible when the crucible is centered in the middle of the grate bars of the present furnace. For in the latter position, if an open seat is used, cold air coming through the grate strikes directly on the bottom of the crucible, and if a seat of some refractory material is used, this is also cooled by the draft and cuts off any direct contact of the heat with the under side of the pot.

Another advantage is secured by removing a weight of about 350 lb. from the grate bars, thus doubling or tripling their life. With the present furnace the grate bars must be taken out and straightened after every 80 or 90 bars are poured. Furthermore, lighter bars could be used and more air space obtained, which is an important consideration. The bottom of the furnace can be made horizontal or slightly inclined, as shown in the sketch. The method of handling the doors should be taken into consideration when selecting either design.

The furnace shown in Fig. 2 has never been tried out in practice, but its feasibility is assured, judging from the action of the furnace now in use. It has never been installed at Rio Plata because the saving that could be effected does not warrant tearing out the present furnace. The cost of new firebrick laid down at the mine is excessive, due to transportation over a long and difficult trail.

A second type of furnace that should work equally as well as the present furnace, is shown in Fig. 3. This is practically the same as the Noble petroleum melting furnace, the only difference being the changes that naturally suggest themselves when using charcoal instead of petroleum. The arrows in the sketch show the approximate path of the center of the flame.

Bullion Mold Platform for Tilting Furnaces.—One of the objections to the use of tilting furnaces for melting gold and silver bullion, has been that the mold had to be rather remote from the lip of the crucible, the distance constantly varying, and spilling of the molten metal was almost unavoidable. To correct this defect, the operators at the Tom Reed

Gold Mines Co., Oatman, Ariz., devised the apparatus illustrated in Fig. 241, which has been in use for some months with entirely satisfactory results. The furnace straddles a pit which facilitates the movement of a platform for holding the mold. Constructed of steel after the manner of the pantograph, the platform framework is fastened by iron rods to the front of the furnace at two points, allowing the framework to move freely up and down, but maintaining a constant distance from the crucible lip. This distance may be adjusted to suit the dimensions

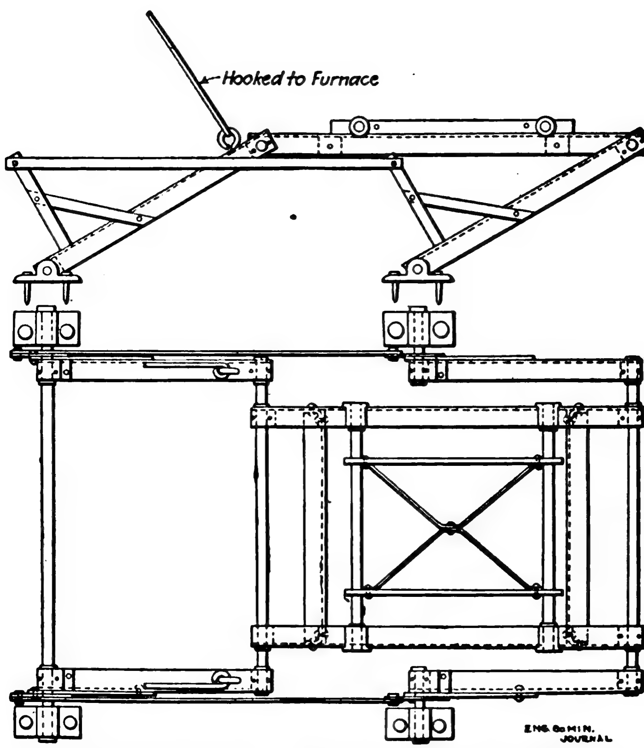


FIG. 241.—MOVABLE MOLD PLATFORM FOR TILTING FURNACES.

of the mold used, which may be either conical or of the rectangular bar type. The small platform which directly receives the mold is on rollers, so that it may be placed more or less directly under the crucible lip as desired. This adjustment is of particular advantage when bars are being poured, when spilling would be costly, although in pouring slag, its advantages are not so marked. Fig. 241 shows in detail the method of construction. It is understood that since its invention at the Tom Reed mine, the appliance is being manufactured and marketed by a supply firm.

A Conical Bullion Mold.—The use of a conical mold for receiving the gold bullion is considered advantageous at the Tom Reed Gold Mines Co., of Oatman, Ariz., for several reasons. One of these is that a bar almost any size can be made. By melting down as much precipitate as the crucible will hold, skimming and then adding more precipitate until the maximum capacity of the crucible is reached, the final bar is made in one melting. Should the weight be more than would be convenient, two bars can be made instead of one. Another advantage of the conical form, and one which receives serious consideration, is that the heavy cone of gold is inconvenient to handle and presents a considerable obstacle to anyone who might try to purloin it. In fact, so clumsy and elusive is the heavy gold cone that it is almost impossible for any one person to pick it up without the assistance of a sling or appropriate tongs.

Annealing Graphite Crucibles.—The fact that graphite crucibles should be carefully annealed before putting them into a hot furnace is often overlooked or not thoroughly understood by operators. It is true, however, that these crucibles are hygroscopic and are likely to carry a considerable percentage of moisture. As they are made under high pressure the contained moisture cannot be driven off rapidly, and if sudden heat is applied, an explosion will result and the crucible will be ruined. This phenomenon is too frequently seen about the melting rooms of metallurgical plants where graphite crucibles are used, and results in an expense too great to be disregarded. On arrival at the plants where they are to be used, these crucibles contain, as a rule, 4 to 5% water. To get rid of this is a necessity, and it must be done slowly. Keeping the crucible stock in the dry heat of the melting room reduces the moisture slowly so that heating a crucible for a few hours on top of the furnace before using it is generally sufficient treatment. A new crucible should never be placed in an old, hot fire unless absolutely sure that it is perfectly dry.

Hints for Graphite Crucible Users.—Some valuable information as to the proper handling of graphite crucibles is given by the Joseph Dixon Crucible Co., Jersey City, N. J., in its catalogue.

Avoid wedging material tightly in crucibles.

Scalping of crucibles comes when one portion of the crucible is heated to a temperature much higher than that of the adjoining portions. The expansion of the more highly heated portion is sufficient to rend it apart from the unexpanded cooler part.

Crucibles may easily absorb 5% of water from the air, and the presence of water in any part of the crucible prevents that part from getting heated above 212 deg. F., so that it is important that the crucible should be both warm and dry before being sharply heated. A small fire inside

of the pot during the night before using, is a simple makeshift for those not having a better arrangement.

In the use of oil fuel, perfect combustion is the end to be achieved. Too little oil or too much steam (or air) brings about an oxidizing condition that extracts the carbon from the crucible wall.

A large excess of air at low red heat is bad, as the carbon of the crucible is then consumed. This tends to make the pot porous, develops "alligatoring," and rapid wasting away.

Do not heat up a new crucible too rapidly.

If the crucibles show excessive cutting or fluxing at one point, it indicates a higher temperature at that point, and that the furnace is not in proper order.

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